BMI/OCRD-4(3)
Distribution Category UC-70

Hard Rock Excavation at the CSM/OCRD Test Site Using Swedish Blast Design Techniques

Technical Report

September 1983

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The content of this report was effective as of July 1983. This report was prepared by the Department of Geological Engineering, Colorado School of Mines under Subcontract E512-04800 with Battelle Project Management Division, Office of Nuclear Waste Isolation and Office of Crystalline Repository Development under Contract Nos. EY-76-C-06-1830 and DE-AC02-83CH10140 with the U.S. Department of Energy.

BIBLIOGRAPHIC DATA

Holmberg, Roger, 1983. *Hard Rock Excavation at the CSM/OCRD Test Site Using Swedish Blast Design Techniques*, BMI/OCRD-4(3), prepared by Swedish Detonic Research Foundation for Colorado School of Mines for Office of Crystalline Repository Development, Battelle Memorial Institute, Columbus, OH.

NOTICE

Printed in the United States of America
Available from
National Technical Information Service
U.S. Department of Commerce
5285 Port Royal Road
Springfield, VA 22161

NTIS price codes
Printed copy: 06
Microfiche copy: A01

ABSTRACT

This report is the third in a series describing research conducted by the Colorado School of Mines for the Office of Crystalline Repository Development (OCRD) to determine the extent of blast damage in rock surrounding an underground opening. A special room, called the CSM/OCRD room, was excavated at the CSM experimental mine for the purpose of assessing blast damage in the rock around the room. Even though this mine is not proposed as a nuclear waste repository site, the instrumentation and methods of blast damage assessment developed in this project are applicable to proposed repository sites.

This report describes the application of Swedish blasting technology for the excavation of the test room. The design of the blasting patterns including the selection of explosives, hole sizes and location, explosive loading densities, and delay intervals is based upon the theories of Langefors and Kihlstrom in combination with methods used at the Swedish Detonic Research Foundation for minimizing unwanted rock damage. The practical application of the design procedures to seven rounds and the achieved results are discussed.

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EXECUTIVE SUMMARY

The Swedish approach to blast design was applied to seven of the rounds used in the excavation of an experimental room at the CSM mine. The basis for the design, the procedures which were followed and the results achieved are discussed in detail. Although the exact nature and extent of the unwanted blast damage will only be determined through future studies, visual observation suggests that the disturbance has been minimal. This technique can be easily modified for application in other hard rock types.

1 INTRODUCTION

The objective of this task was to demonstrate how Swedish blasting design techniques might be used in the construction of an actual nuclear waste repository in crystalline rock. Special attention was focused on minimizing damage to the surrounding rock. The actual demonstration of these methods resulted in the excavation of a test room 30 m long, 4.5 m wide, and 3 m high at the Colorado School of Mines (CSM) Experimental Mine located at Idaho Springs, Colorado. The Swedish blasting design method based upon theory developed by Langefors-Kihlstrom [1] was used to design seven rounds. These rounds were blasted during August, 1979. Another three rounds were designed by P.A. Sperry, a consultant to CSM, based upon the Livingston crater theory. These latter results will be presented in the fourth technical report in this series.

This report outlines the Swedish methods for blasting design and gives the result from the blasting of the seven rounds in the test room. A preliminary estimate of the extent of blasting damage in the surrounding rock is included.

2 THE SWEDISH APPROACH TO TUNNEL BLAST ROUND DESIGN

2.1 INTRODUCTION

When extending a tunnel in rock, there is initially only one surface (the end of the tunnel) towards which to break the rock. If one could drill holes parallel to this free surface, then the rock ahead of the face (end of the tunnel) could be slabbed (broken) off. Normally, however, one must fragment the rock through the use of parallel holes drilled perpendicular to the end of the room. In this case, the first step is to create a cut (slit) in the rock to the depth of the round, thereby providing additional surfaces towards which subsequent breakage can occur. The second free face is initiated by drilling one or more holes in close proximity in the central portion of the face. This hole (holes), which will not be loaded with explosives, is often larger than the others in the round. In this discussion, it will be assumed that one empty hole of diameter $\emptyset$ will be used. The diameter of the blastholes is d (where $d < \emptyset$). Once the size of the empty hole has been determined, the nearest blastholes (Figure 2-1) are located (distance V_1) such that the interlying rock can be cleanly broken and ejected into the tunnel. The amount of explosive energy used per unit length of hole (charge concentration) depends upon the type of explosives available, the type of rock, the location of the holes with respect to the free surface, and the degree of confinement (i.e., the rock in the corner of a room is harder to break than that near the center). Once the holes closest to the empty hole have detonated the next closest holes (distance of V_2 from the slot, Figure 2-2) are initiated. This second group of holes can be located further from this slot than the first ($V_2 < V_1$). The process continues until the stoping holes can take over. The most common type of cut used in Sweden is the parallel hole cut (Figure 2-3) where quadrangles of holes are centered around an empty hole. The diameter of the empty hole is larger than the holes in the round.

The next group of holes are termed stoping holes (Figure 2-4) as they have a relatively large surface toward which to break. Several holes can be positioned along one side of the cut as opposed to just expanding the cut by one hole per side.

As one approaches the periphery of the desired opening, particular attention must be placed upon designing patterns which will minimize damage to the surrounding rock. Although one normally is concerned about the contour holes because of opening stability considerations, in this application (where the waste may be stored in the floor), the lifters (floor holes) must also be carefully designed.

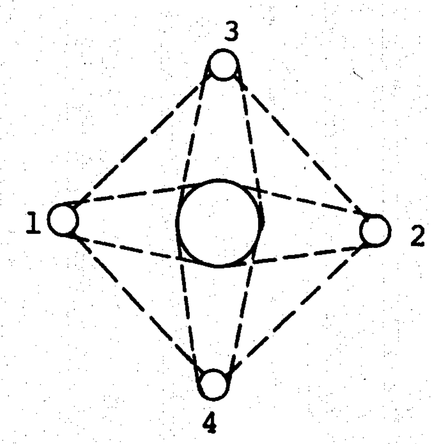


Figure 2-1. Initiation Sequence for Quadrangle 1

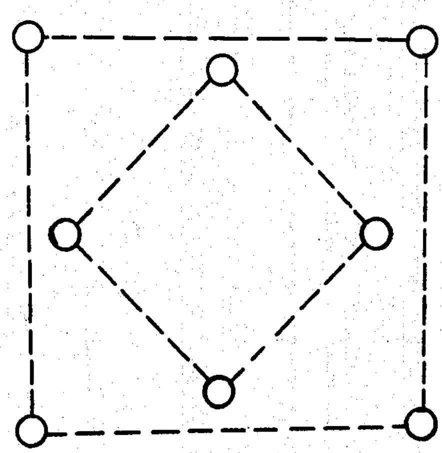


Figure 2-2. Idealized Representation of Quadrangle 2

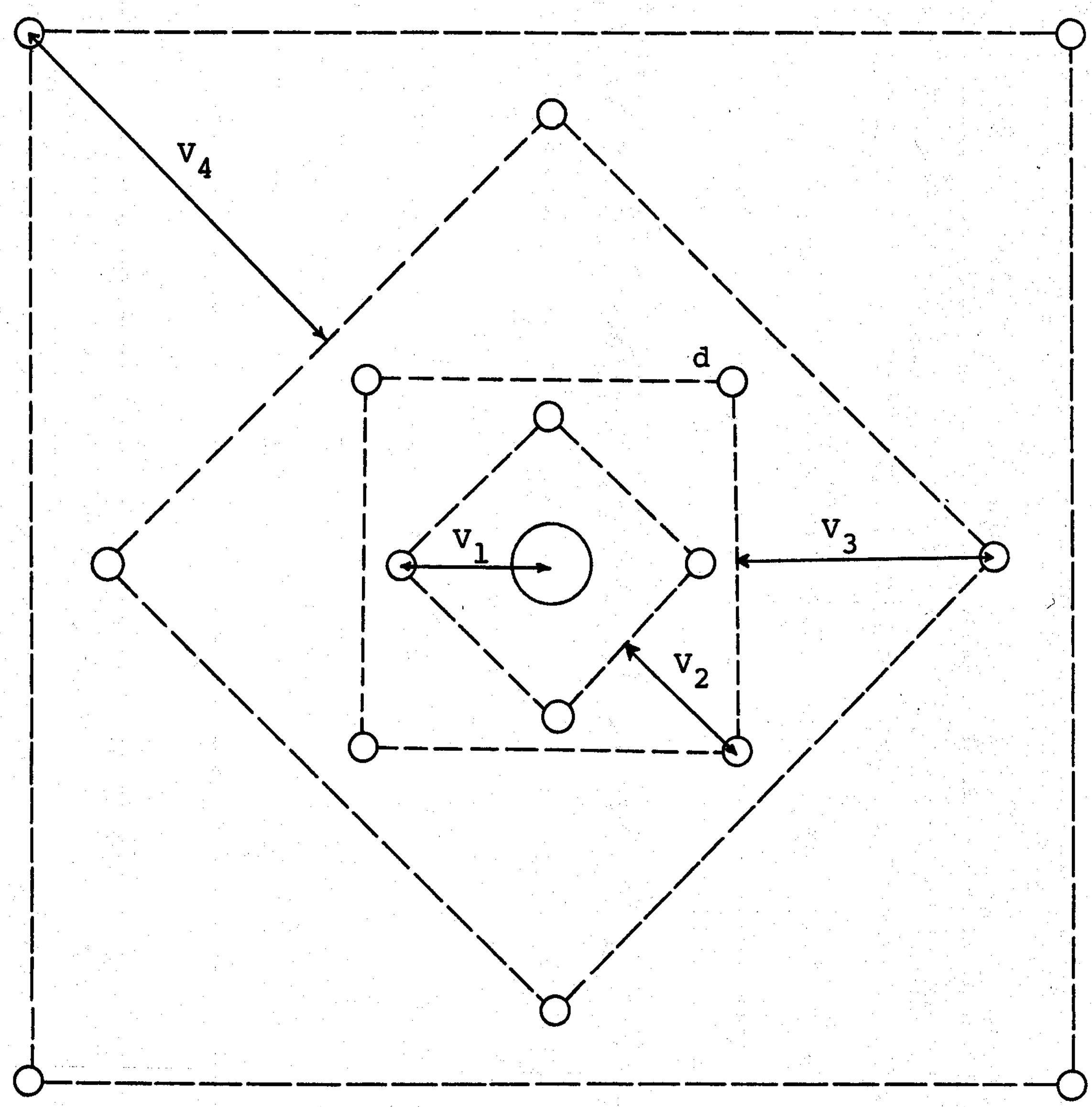


Figure 2-3. Four Section Cut

NOTE: V_1 Represents the Different Practical Burdens for Quadrangle No. 1.

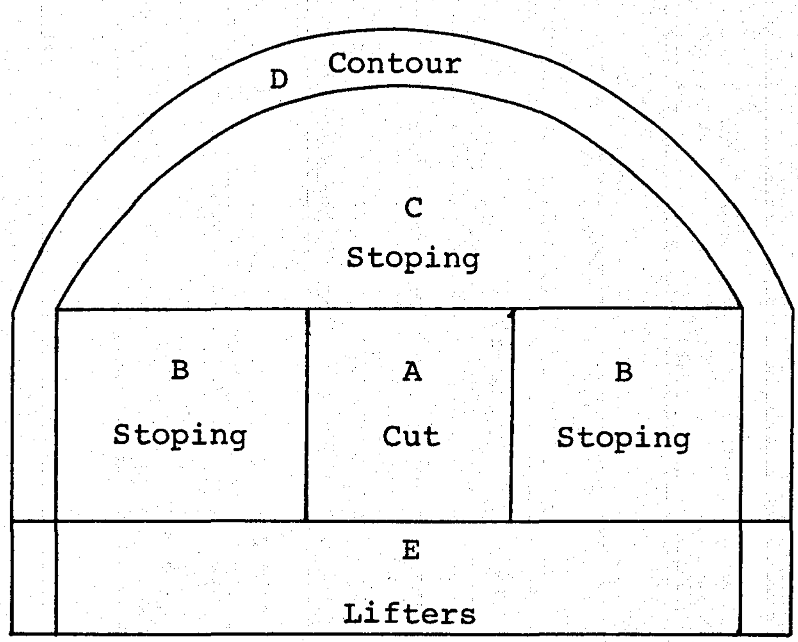


Figure 2-4. Nomenclature Used to Describe the Different Parts of a Tunnel Round

The spacing of the contour holes and lifters is designed such that upon detonation, crack formation is encouraged between holes and towards the free face. The amount and type of explosive and the burden (distance to the free surface) must be carefully controlled. As will be shown later, the next inner row of holes (those towards the cut) can, if not carefully controlled, produce unwanted damage to the rock outside of the projected tunnel periphery. Therefore, care must be taken in designing the hole spacing, charge density, and burden of these as well.

The face area remaining between the two rows of contour/lifter holes and the cut are filled as needed with stoping holes.

It is clear that a design is often one thing and the execution of the design is something else. If the holes are not in the correct position, then the rock will not be blasted as intended. The potential problems fall into three categories:

1) collaring errors (the holes are not drilled at the proper x, y face coordinate),
2) hole deviation (the holes are not drilled at the proper angle with respect to the face), or
3) improper loading of the holes, improper detonation sequencing, etc.

The occurrence of problems 1 and 2 means that the actual values for burden and spacing will be different from those used to calculate the amount of required explosive. Problem 3 means that the holes are improperly charged or sequenced to do the work required. For all three, poor blasting results will be experienced. All can be controlled/minimized with careful supervision, well trained personnel, and good equipment. Even so, collaring errors and hole deviation will be present and these must be acknowledged in both the design and the practice. They are included in the design by reducing somewhat the calculated hole burdens and spacing. These reduced values are termed "practical values". If the actual hole deviation and collaring errors exceed those included in the practical calculations then adjustments must be made in the field (i.e., redrilling of holes and/or adjusting charge densities).

2.2 CHOICE OF BLASTING AGENTS FOR EXCAVATION

The most important parameters when using careful blasting procedures to drive a drift or a tunnel in hard rock are the drilling accuracy and the behavior of the blasting agent. If too little attention is placed on the drilling, an irregular contour and a greater disturbance in the surrounding rock mass will be the result. A hole that is drilled outside of the planned contour has a much higher degree of fixation (confinement). This makes it harder to break the rock between rows (burden) and produces unwanted cracking which affects the stability of the opening.

A suitable explosive must have the ability to detonate with various degrees of decoupling* so that the charge concentration (explosive energy per length of borehole) can easily be changed depending on where in the round the charge is to be placed. In order to keep the number of drill holes to a minimum, stoping in the center of the round requires a high charge concentration. On the other hand, stoping close to the perimeter requires a charge concentration that does not affect the remaining rock more than smooth wall blasting.

For a number of explosives, channel effects cause detonation failures if the coupling ratio becomes much less than one. Channeling occurs when the expanding detonation gas compresses the air in the annulus (channel) between the charge and the borehole wall forming a high temperature and high pressure layer. The shock front in the air compresses the explosive in front of the detonation front and destroys the hot spots or increases the density to such a degree that the detonation process could stop or result in a low energy release. This occurs mainly for explosives with detonation velocities less than about 3,000 m/s.

Explosives used in the lifters must be able to withstand water. At the contour there is a need for a low charge concentration explosive to minimize damage to the remaining rock. This is particularly true for nuclear waste storage applications where the increase in permeability due to excavation must be kept to a minimum.

For any given borehole, the quantity of explosive needed per meter of borehole (ℓ) depends upon the weight density (kgf/m^3) and the weight strength (energy content/kgf). The required weight of explosive per meter would for example be different when using an ammonium nitrate-fuel oil (ANFO) mixture than when using dynamite.

In Sweden, the weight strength of an explosive is expressed relative to a standard dynamite-based explosive designated LFB. The formula for relative weight strength is

$$s = \frac{5}{6}\frac{Q}{Q_o} + \frac{1}{6}\frac{V}{V_o} \qquad (2\text{-}1)$$

---

* A fully coupled explosive is one which completely fills the borehole. As the diameter of the explosive is reduced compared to that of the borehole, the coupling decreases.

where

s = weight strength relative to a reference explosive (LFB-dynamite)

Q_o = heat of explosion for 1 kg of LFB (5 MJ/kg)

V_o = released gas volume at standard temperature and pressure (STP) from 1 kg of LFB ($0.85 m^3/kg$)

Q = heat of explosion for 1 kg of the actual explosive

V = released gas volume (STP) from 1 kg of the actual explosive

An explosive for which s = 1.5 would mean that it contains 1.5 times as much breaking power per unit weight as LFB.

The weight strength of an explosive can be expressed relative to ANFO by first calculating the weight strength relative to LFB and then dividing by the weight strength of ANFO relative to LFB (0.84). Representative values for several explosives are given in Table 2-1.

Table 2-1. Weight Strength for Some Explosives

Explosive	Q MJ/kg	V M^3/kg	s LFB	s ANFO	Density kg/m^3
LFB Dynamite	5.00	0.850	1.00	1.19	
Dynamex B	4.6	0.765	0.92	1.10	1450
ANFO	3.92	0.973	0.84	1.00	900
TNT	4.1	0.690	0.82	0.98	1500
PETN	6.12	0.780	1.17	1.39	
Nabit	4.1	0.892	0.86	1.02	1000
Gurit	3.73	0.425	0.71	0.85	1000

2.3 EVALUATION OF ROCK BLASTABILITY

In the formulas of Langefors and Kihlstrom, the amount of explosive required to loosen a cubic meter of rock in a specified geometry is expressed in terms of a rock constant, c. This is an empirical constant, some values of which are given in Table 2-2.

Table 2-2. Rock Constants for Various Rock Types

Rock Type	c Value
Brittle Crystalline Granite	0.2 kg/m^3
Most Other Rocks	0.3-0.4
Most Swedish Granite	0.4 kg/m^3

Under Swedish conditions c = 0.4 kg/m^3 is used predominantly.

2.4 PRINCIPLES OF TUNNEL ROUND DESIGN

This section presents a very brief description of the formulas used to design the layouts for different tunneling rounds. A much more detailed description of the method can be found in Appendix A. The principles upon which the calculation method is based are described in the book, The Modern Technique of Rock Blasting by U. Langefors and B. Kihlstrom.(1)

2.4.1 Advance

The length of the round which can be blasted (advanced) at one time is determined primarily by the diameter of the empty hole and the hole deviation. The relationship between hole depth and empty hole diameter for the typical case of 95 percent advance (of the drilled depth) using a parallel hole cut is shown in Figure 2-5.

For an empty hole diameter of 89 mm (3.5 in), the maximum hole depth should be about 2.8 m. The expected length of the round pulled would then be 2.8 x 0.95 = 2.66 m.

The equation of the curve in Figure 2-5 can be expressed as

$$H = 0.15 + 34.1\,\emptyset - 39.4\,\emptyset^2 \qquad (2\text{-}2)$$

where

$\emptyset$ = diameter of the empty hole (m)

H = drilled depth of the hole (m)

The general geometry for the design cut is shown in Figure 2-3. A good rule of thumb for deciding the number of quadrangles in the cut is that the side length of the last quadrangle should not be less than the square root of the advance. If, for example, the advance is 2.66 m, then the side length of the last quadrangle should not be less than $\sqrt{2.66}$.

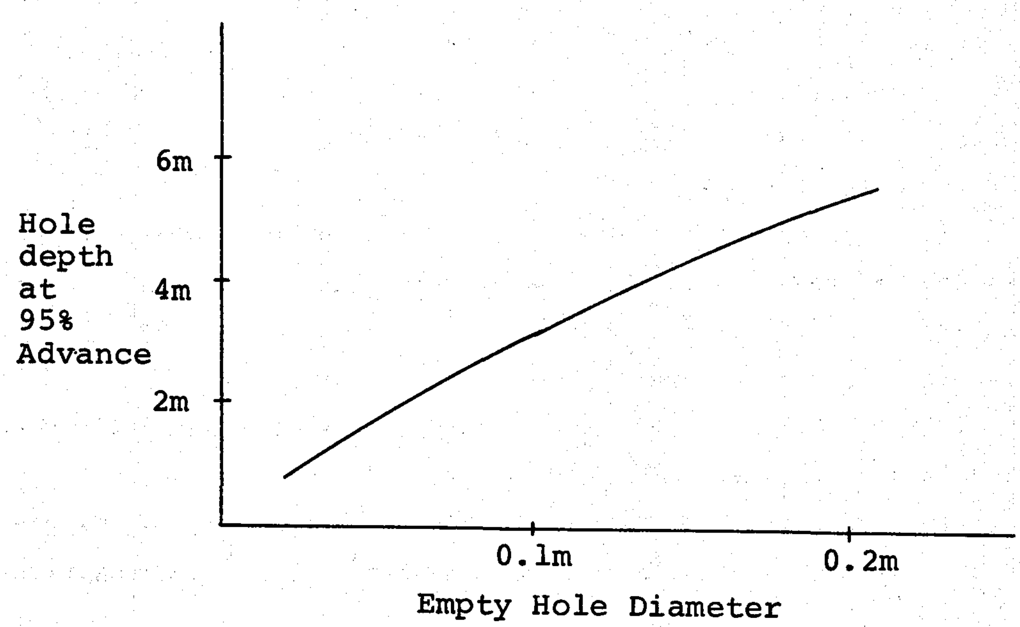


Figure 2-5. Hole Depth as a Function of Empty Hole Diameter for a Parallel Hole Cut (Four Section Cut)

2.4.2 Cut

The burden for the first quadrangle should never exceed 1.7 times the diameter of the empty hole if satisfactory breakage and cleaning is to take place. In practice, hole deviations reduce this value. For a hole deviation of the magnitude of 1%, the burden should be reduced to

$$V_1 = 1.5\ \emptyset \tag{2-3}$$

where

V_1 = practical burden (m) of first quadrangle

$\emptyset$ = diameter of empty hole (m).

The amount of explosive that should be used in the holes making up the first quadrangle is

$$\ell_1 = 55\ d\ (V_1/\emptyset)^{1.5}\ (V_1-\emptyset/2)\ (c/0.4)/s_{ANFO} \tag{2-4}$$

where

ℓ_1 = charge concentration (kg/m) required of the selected explosive

d = diameter of blastholes (m)

s_{ANFO} = weight strength of the explosive used relative to ANFO

c = rock constant (normally c = 0.4 kg/m^3).

The charge concentration required when blasting towards an empty circular hole is naturally higher than that required in the remaining quadrangles of the cut due to the high construction and less effective stress wave reflection.

Once the first quadrangle has been removed (Figure 2-1), the holes in quadrangles 2, 3, etc., will be blasted towards a straight face of length, B. This distance is that which existed between the previous holes.

By knowing the charge concentration, ℓ, for the planned explosive and the width, B, the burdens for the remaining quadrangles can be calculated using equation 2-5.

$$V = 8.8\ 10^{-2} \left(\frac{\sqrt{2}(B-F)\ell\ s_{ANFO}}{dc}\right)^{1/2} \tag{2-5}$$

where

B = width of the theoretical free face (m)

F = faulty drilling correction (m) = $\gamma H + \psi$

γ = angular hole deviation from correct position (m/m)

H = hole depth (m)

ψ = collaring error (m)

ℓ = charge density (kg/m) of the selected explosive.

A restriction on V is that it should be less than twice the opening, B, to prevent plastic deformations. It is noted that the positioning of the holes follows naturally since they are at the corners of quadrangles rotated through 45° from the previous quadrangle.

2.4.3 Stoping Holes and Lifters

The burdens for the lifter and stoping holes should be calculated using the following formula:

$$V = 0.9 \quad \left(\frac{\ell \quad s_{ANFO}}{(c + 0.05)\ f\ (E/V)}\right)^{1/2} \qquad (2\text{-}6)$$

where

f = fixation factor

E/V = hole spacing to burden ratio

E = hole spacing (m)

V = burden (m) .

In the formulas, different fixation factors, f, are used for calculating the burden in different situations. For example, f=1 in bench blasting with vertical holes positioned in a row with a fixed bottom. If the holes are inclined, it becomes easier to loosen the toe. To account for this, a lower fixation factor ($f<1$) is used for an inclined hole. This results in a larger possible burden. In tunneling, a number of holes are sometimes blasted with the same delay number. Sometimes the holes have to loosen the burden upwards and sometimes downwards. To include the effects of multiple holes and of gravity, different fixation factors are used.

The fixation factor and the E/V relationship depends upon whether the holes are lifters or are holes for horizontal or vertical stoping. For stoping holes breaking horizontally and upwards (see Figure 2-4) a fixation factor (f) of 1.45 and an E/V ratio equal to 1.25 is used. The fixation factor for stoping holes breaking downwards is reduced to 1.2 and E/V should be 1.75. For lifters, f = 1.45 and E/V = 1.

2.4.4 Contour Holes

If smooth blasting is not necessary, formula 2-6 could be used for the contour holes as well. Smooth blasting in an average Swedish bedrock reveals that the desired spacing, E, is a linear function of the hole diameter.

$E = k_1 d$ (2-7)

where the constant k_1 is of the order of 15. An E/V ratio of 0.8 is predominantly used. For a 41 mm hole diameter, the spacing will be about 0.6 m and the burden aboout 0.8 m.

In practice, the perimeter holes are not drilled parallel to the others in the round, but rather are inclined outward from the centerline of the tunnel. The amount of this "look-out" (difference in end positions between the inclined hole and the stoping holes) must be included in computing the burden for the contour holes.

The minimum charge concentration per meter of borehole is also a function of the hole diameter (Figure 2-6). For hole diameters up to 25.4 cm, the relationship is given below.

$\ell = 90\ d^2$ kg/m (2-8)

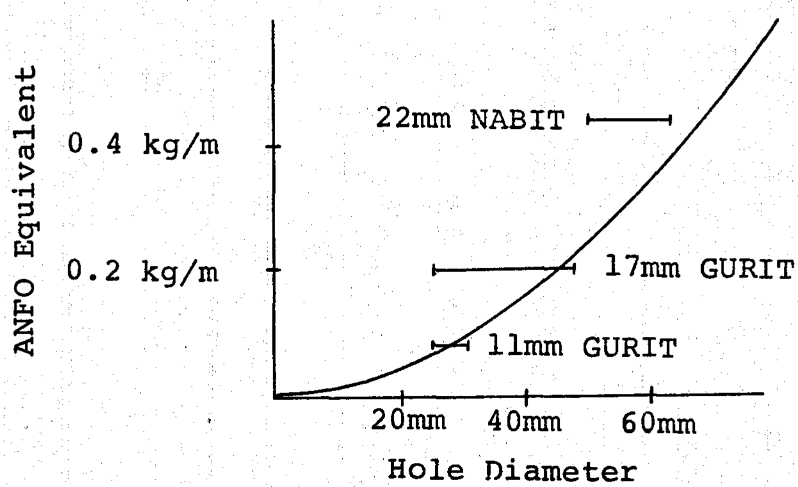


Figure 2-6. Minimum Required Charge Concentrations and Recommended Hole Diameters for Smooth Wall Blasting with Some Swedish Explosives

2.5 CAREFUL BLASTING AND BLAST DAMAGE

When an explosive charge detonates in a borehole, the expansion of the high pressure gaseous reaction products sets the borehole walls in an outward motion, creating a dynamic stress field in the surrounding rock. The initial effect in the nearby rock is a high intensity, short duration shock wave which quickly decays. The gas expansion leads to further motion and sets up an expanding stress field in the rock mass. The rock motion is strongly dependent on the presence of nearby free surfaces. When the free surface is close enough, the rock breaks free. In directions other than towards the free surface, the motion spreads in the form of the well-known ground vibration waves. These are a complicated combination of elastic waves in which the rock reverberates in the compressive, P, shear, S, and surface, R, wave modes. Each mode or wave has a characteristic propagation velocity, C, that is some fraction of the sonic velocity (a material property of the rock mass). The particles in the rock mass move through an approximately elliptical path with the peak particle velocity, v, decreasing with the distance away from the charge. Damage is a result of the induced strain, ε, which for an elastic medium using the sine-wave approximation, is given by the relation:

$$\varepsilon = v/C \tag{2-9}$$

In the region close to the charge, permanent damage occurs at a given critical level of particle velocity. The degree to which the damage affects the stability (stand-up time) of the rock contour depends upon the nature of the damage, the rock structure, the ground water flow, and the orientation of the damage planes in relation to the contour and the existing (static) load.

During a blast there are, of course, different types of damage that occur. In the region closest to the charge, crushing occurs if the compressive stress exceeds the compressive strength, radial cracks appear, and tensile stresses due to reflection waves open existing micro cracks. The shear strength of existing joints is reduced due to movement or opening of the joints. Damage due to compression waves is of minor danger to the rock stability. The radial cracks, however, may affect the stability. Their length is dependent upon the distances between joints in the rock mass. They propagate until the stress concentration factor becomes too low or until the crack hits a joint. If a joint is nearby, the detonation gases penetrate into the joints and thereby lower the shear strength. Joints or closed cracks in the surrounding rock mass exhibit wide strength variations. The vibrations caused by the detonations decrease in amplitude with distance. Due to the different strengths of the joints in the vicinity, only some of them are affected.

At the Swedish Detonic Research Foundation (Sve De Fo), a rock mass damage (with regard to local stability) criterion has been developed on the basis of: a) changes in the number of cracks observed in cores taken from the surrounding rock mass prior to and after blasting; b) rock displacement measurements; and c) peak particle velocity measurements made using accelerometers in regions very close (less than fifteen meters from 25.4 cm holes and less than two meters from 3.8 cm holes) to the detonating charges. Such measurements have encompassed a range of rock types.

The mathematical model for predicting peak particle velocities described in this section is in good agreement with measured values in granite and gneiss. For these rock types, the damage threshold appears to be between 700-1,000 mm/sec. This model has been used for predicting the rock damage at the CSM Experimental Room and for optimizing the design of the blasting patterns.

For an extended charge of linear charge concentration, ℓ, a first approximation of the resulting peak particle velocity, v, can be obtained by integrating equation (2-10) with respect to the position along the charge.

$$v = k\, W^{\alpha}/R^{\beta} \tag{2-10}$$

where

v = peak particle velocity in mm/sec.

W = charge weight in kg

R = distance in m

k, α and β = constants.

From experimental and theoretical considerations, one assumes that the effective parts of the elemental waves arrive at point A (Figure 2-7) almost simulatneously. The difference in time of arrival of the elemental waves from different parts of the charge is neglected. The distance to the point of observation A is given by

$$R_i^{\,2} = R_o^{\,2} + (R_o \tan\theta - x_i)^2 \tag{2-11}$$

where

R_o = the perpendicular distance from the charge to the point of observation A

θ = the elevation angle to point A

x_i = the distance from the end of the charge to the elemental charge W_i

$$W_i = \ell\, dx\,. \tag{2-12}$$

By integrating equation (2-10) over the charge length, H, the peak particle velocity can be calculated from

$$v = k\, \ell^{\alpha} \int_0^H \frac{dx}{\left[R_o^{\,2} + (R_o \tan\theta - x)^2\right]^{\beta/2\alpha}}\,. \tag{2-13}$$

For one special explosive in competent Swedish bedrock the constants are $k = 700$, $\alpha = 0.7$, and $\beta = 1.5$. For an arbitrary explosive, the charge concentration, ℓ, in equation (2-13) must be normalized with respect to this explosive which has a weight strength of 1.02. The weight strength of any explosive relative to ANFO is given by

$$s_{ANFO} = \frac{Q + V_g/0.85}{5.04} \tag{2-14}$$

where Q denotes the heat of explosion in MJ/kg and V_g denotes the released gas volume at STP, m^3/kg.

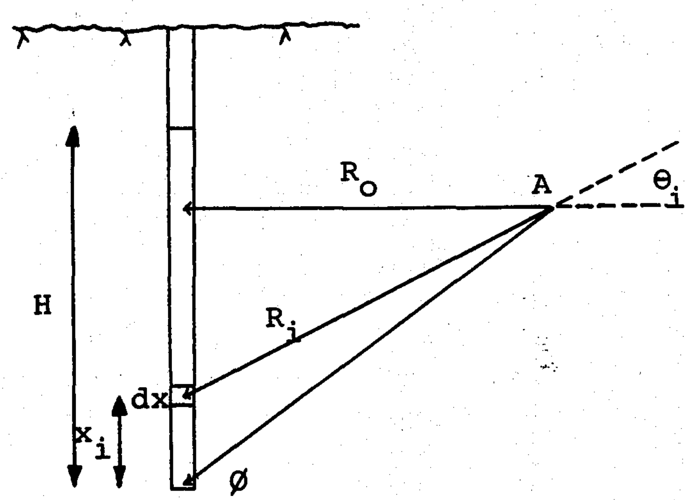


Figure 2-7. Integration Over Charge Length to Calculate Particle Velocity at an Arbitrary Observation Point

This means that for the actual explosive the charge concentration, ℓ, to be used in equation (2-6) should be determined according to equation (2-15).

$$\ell = \ell_{\text{explosive used}} \times \frac{s_{\text{ANFO}}}{1.02} \tag{2-15}$$

Figure 2-8 presents calculated curves of v as a function of R (the perpendicular distance to the extended charge) with the linear charge density, ℓ, as a parameter, for a 3-m long charge. For a charge concentration of about 1.0 kg/m the extent of the damage zone (peak particle velocity greater than 1,000 mm/sec) is predicted to be about 1.1 m.

For the Experimental Room, the velocity expected to give incipient fracture has been estimated as 800 mm/s. Using the theory described above, the extent of the damage zones can be estimated for the different charges used in the blasting excavation. These results are summarized in Table 2-3.

Table 2-3. Estimated Damage Zones for the Different Explosive Concentrations

Explosive	Charge Length (m)	Charge Conc. (kg/m)	Weight Strength (s_{ANFO})	Damage Zone (m)
Tovex 100	2.0	0.56	0.85	0.74
Tovex 210	2.0	0.80	0.92	0.96
Tovex 220	2.0	0.99	0.93	1.08
PETN 2x200	2.4	0.08	1.39	0.28
3x200	2.4	0.12	1.39	0.36
4x200	2.4	0.16	1.39	0.45

In designing the rounds, the predicted damage to the surrounding rock from the stoping holes should not exceed the damage zone caused by perimeter holes. It must be pointed out, however, that the predicted values of the damage zone could be exceeded if the borehole is greatly confined. This usually happens if borehole deviation is bad.

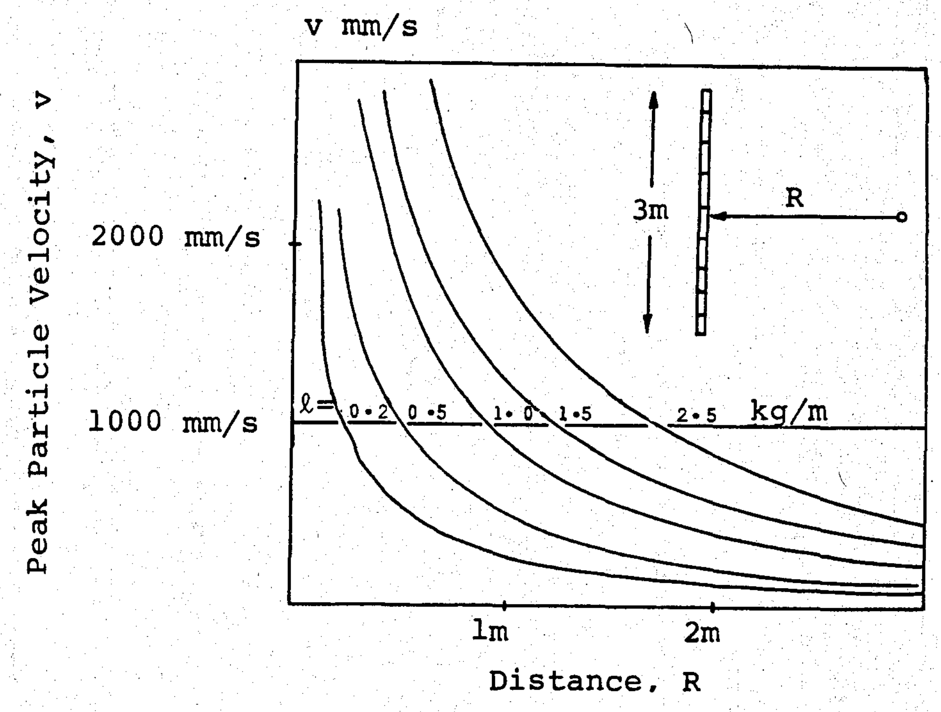


Figure 2-8. Estimated Peak Particle Velocity as a Function of Distance for Different Linear Charge Densities

3 SITE FOR EXCAVATION OF EXPERIMENTAL ROOM

3.1 LOCATION AND GEOLOGY

The location within the CSM Experimental Mine selected for the experimental room is shown in Figure 3-1. The bearing of the room is S23E, and the room is positioned between A-Left spur and Miami tunnel with the entrance from A-Left.

Before beginning the design of a blasting program, the available information regarding the rock types and structures should be collected and reviewed. The principal rock formations in the Idaho Springs area are of Precambrian age. Some of them have undergone severe metamorphism, during which a schistose or gneissic structure has been developed. The oldest and most extensive formation is the Idaho Springs formation which is predominantly a biotite schist. These rocks were cut extensively by porphyritic dikes of Tertiary age, predominantly monzonite porphyry. Subsequent folding and shearing allowed the intrusion of Precambrian igneous rocks. The oldest igneous intrusions are silivous pegmatites and hornblende gneisses. Successive intrusions of quartz-monzonite, granite, more quartz monzonite diorite, and, finally, a distinct type of biotite granite followed.

Faults of small displacement are quite frequently observed. Movements occurring after the solidification of the porphyry caused fissures to form. These fissures were more or less filled by loose fragments of the porphyry and country rock. Subsequent to the cementing of the brecciated fragments by ore deposition in the fissures, faulting was renewed along the same fractures. Due to this secondary faulting, the vein matter was compacted into a secondary breccia mixed with quartz. Comparatively recent movements have, in turn, deformed the resulting vein faulting, and as a result, the veins contain fragments of ore, quartz, and country rock in a loose matrix of finely crushed material. Many of these movements along the vein fissures have been of considerable displacement as is evident by the well rounded fragments in some veins.[(2)]

3.2 FRACTURE ORIENTATION ON SITE

A preliminary fracture analysis of the mining faces in the experimental room has been done by P. Rosasco, a graduate student at CSM. Although not as yet completely analyzed, a compilation of 710 fracture orientations obtained by mapping of mining faces reveals at least 20 individual joint sets. The strikes and dips of these joints sets are given in Table 3-1 and Figure 3-2.

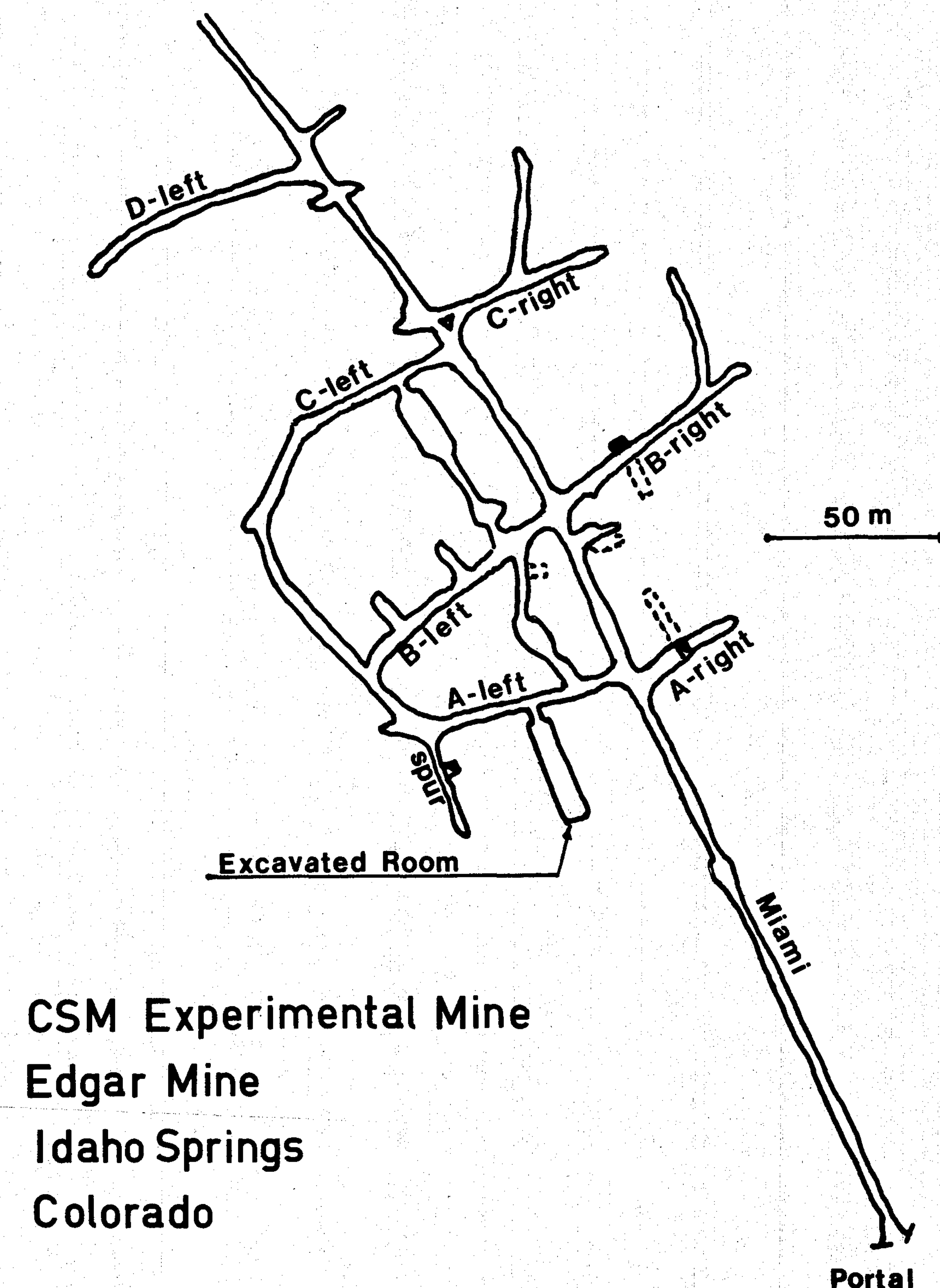


Figure 3-1. Plan View of that Part of the CSM Experimental Mine Containing the CSM/OCRD Test Facility

Table 3-1. Orientation of Joint Sets Measured from Contour Diagrams on Schmidt Equal Area Plots of Fracture Poles

Joint Set No.	Strike and Dip
1	N69E, 72NW (metamorphic foliation)
2	2N542, Vertical
3	N45E, 57SE
4	N32E, 45SE
5	N68W, 71NE
6	N41W, 83NE
7	N24W, Vertical
8	N45W, 50SW
9	N7W, 30NE
10	N78W, 56SW

The bearing of the experimental room is S23E.

Data collected by P. Rosasco, a CSM graduate student.

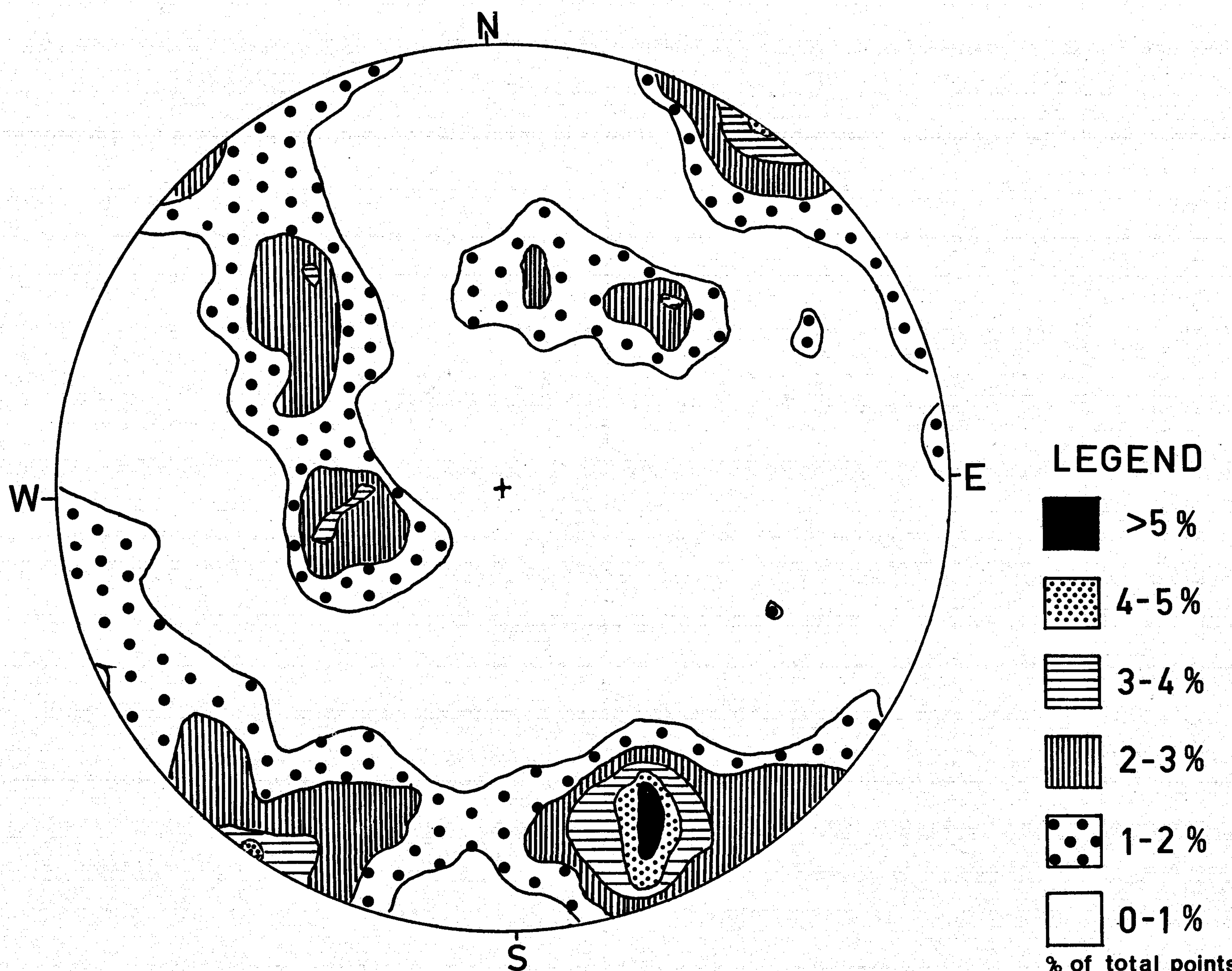


Figure 3-2. Contour Diagram on Lower Hemisphere Schmidt Equal-Area Net of 710 Fracture Orientations from the Mapping of Mining Faces

4 APPLICATION OF THE SWEDISH BLAST DESIGN METHOD AT THE CSM MINE

4.1 EXPLOSIVE SELECTION

In section 2, the principles of blast round design were discussed. One of the essential parameters was the explosive charge concentration per unit length, ℓ, of hole. For preventing unwanted damage to the surrounding rock, it is most important that a proper charge concentration be achieved.

For the purpose of selecting the best explosive, data sheets for various dynamites, water gel explosives, and emulsions were obtained from the various manufacturers. Unfortunately, little information was available regarding the

- heat of formation
- gas volume release

from which to estimate weight strength or data on

- critical diameter
- detonation velocity

as a function of the explosive and borehole diameter.

Furthermore, it was difficult to get special explosives in small quantities for testing at the mine. As a result, the explosives used were selected from locally available stock.

The dimensions and specifications of the aluminized water-gel used for the cut and the stoping holes are shown in Table 4-1.

Table 4-1. Specifications of Explosives Used for Drifting

Explosive	ϕ(mm)	ℓ(kg/m)	Q(MJ/kg)	V(m^3/kg)
Tovex 100 (1" x 16")	25	0.56	3.2	0.90*
Tovex 210 (1 1/8" x 16")	29	0.80	3.65	0.85*
Tovex 220 (1 1/4" x 16")	32	0.99	3.7	0.78*

For the contour holes, it was desired to use an explosive having a charge concentration of not more than 0.20 kg/m (0.13 lb/ft). Since the available commercial products

Atlas Powder: Kleen Kut H; ℓ = 0.39 kg/m (0.26 lb/ft)

DuPont: Tovex T-1; ℓ = 0.37 kg/m (0.25 lb/ft)

had values considerably higher than desired, another solution had to be found.

It was possible to obtain from Ensign Bickford a 200 grain (ℓ = 0.042 kg/m) PETN-cord that was suitable for our purposes.

The heat of explosion, Q, for PETN is 6.12 MJ/kg and the released gas volume, V, is 0.89 m^3/kg. The calculated weight strengths for the selected explosives relative to LFB and ANFO are given in Table 4-2.

Table 4-2. Weight Strengths for the Explosives Used

Explosive	s_{LFB}	s_{ANFO}
Tovex 100	0.71	0.85
Tovex 210	0.77	0.92
Tovex 220	0.78	0.93
PETN-cord	1.17	1.39

DuPont MS and Accudet Mark V electric caps were chosen for ignition and delaying of the different charges in the round. Table 4-3 gives the nominal firing times and the actual times that were measured with the geophones used for vibration measurements. The measured firing times are given as the average value of the first and last initiated charge for a given delay number. As can be seen, the only overlap occurs in the No. 13 MS and the No. 1 Accudet delays. This information is important in planning the delay sequence.

4.2 DRIVING OF THE EXPERIMENTAL ROOM

Drilling was performed using jacklegs and a single boom Ingersoll Rand jumbo. Mucking was accomplished using a Bobcat rubber tired loader and an Eimco 12B overshot loader. Rail haulage was employed.

A normal schedule for blasting a round would be:

1. The pattern was designed.
2. A 35 mm slide was made of the pattern. This was projected onto the face of the drift and the hole locations were marked with paint.
3. Drilling with the jacklegs and the jumbo took place with a CSM student assisting in aligning the drill holes.
4. After the drilling was completed, the holes were surveyed. Hole depths, hole deviations, and coordinates were determined for the holes.

Table 4-3. Nominal and Measured Firing Times for DuPont MS and Accudet Mark V Electric Caps

Delay No.	Nominal Firing Time (Sec.)	Measured Firing Time (Sec.)
1 MS	0.025	-
3	0.075	-
5	0.125	-
7	0.175	0.17 ± 0.01
9	0.250	0.25 ± 0.01
10	0.300	0.28 ± 0.01
11	0.350	0.35 ± 0.01
13	0.450	0.46 ± 0.02
1 Accudet	0.5	0.49 ± 0.02
2	1.0	1.01 ± 0.10
3	1.5	1.66 ± 0.09
4	2.2	2.31 ± 0.18
5	3.0	3.14 ± 0.31
6	3.8	4.13 ± 0.21
7	4.6	4.91 ± 0.14
8	5.5	5.67 ± 0.30
9	6.4	6.77 ± 0.22
10	7.4	7.56 ± 0.42
11	8.5	9.12 ± 0.24
12	9.6	10.06 ± 0.21

5. The round was loaded and sand-filled paper bags were used as stemming (packing).
6. Vibration measurements were done during blasting.
7. The round was mucked out. (Hand mucking sometimes had to be done to clean up the floor.)
8. Bootlegs (unblasted ends of drilled holes remaining in the face) were surveyed.
9. Photos were taken and a visual observation of the damage was made.

4.3 BLAST DESIGN

4.3.1 General Information

The preliminary blast round design was made on the following basis:

- 102 mm diameter empty hole (ϕ = 102 mm)
- 45 mm diameter blast holes (d = 45 mm)
- Hole length of 3 meters (H = 3 m)

Due to drilling constraints, the following was actually used:

- 89 mm diameter empty hole (ϕ = 89 mm)
- 38 mm diameter blast holes (d = 38 mm)
- Hole length of 2.4 meters (H = 2.4 m)

During the execution of the excavation program it was desired to vary the patterns in a controlled way and to observe the results. Only visual observations of the blasting success could be made while drifting was underway. The extent of blast damage to the surrounding rock is to be evaluated later using various quantitative techniques. These results will be described in subsequent technical reports in this series.

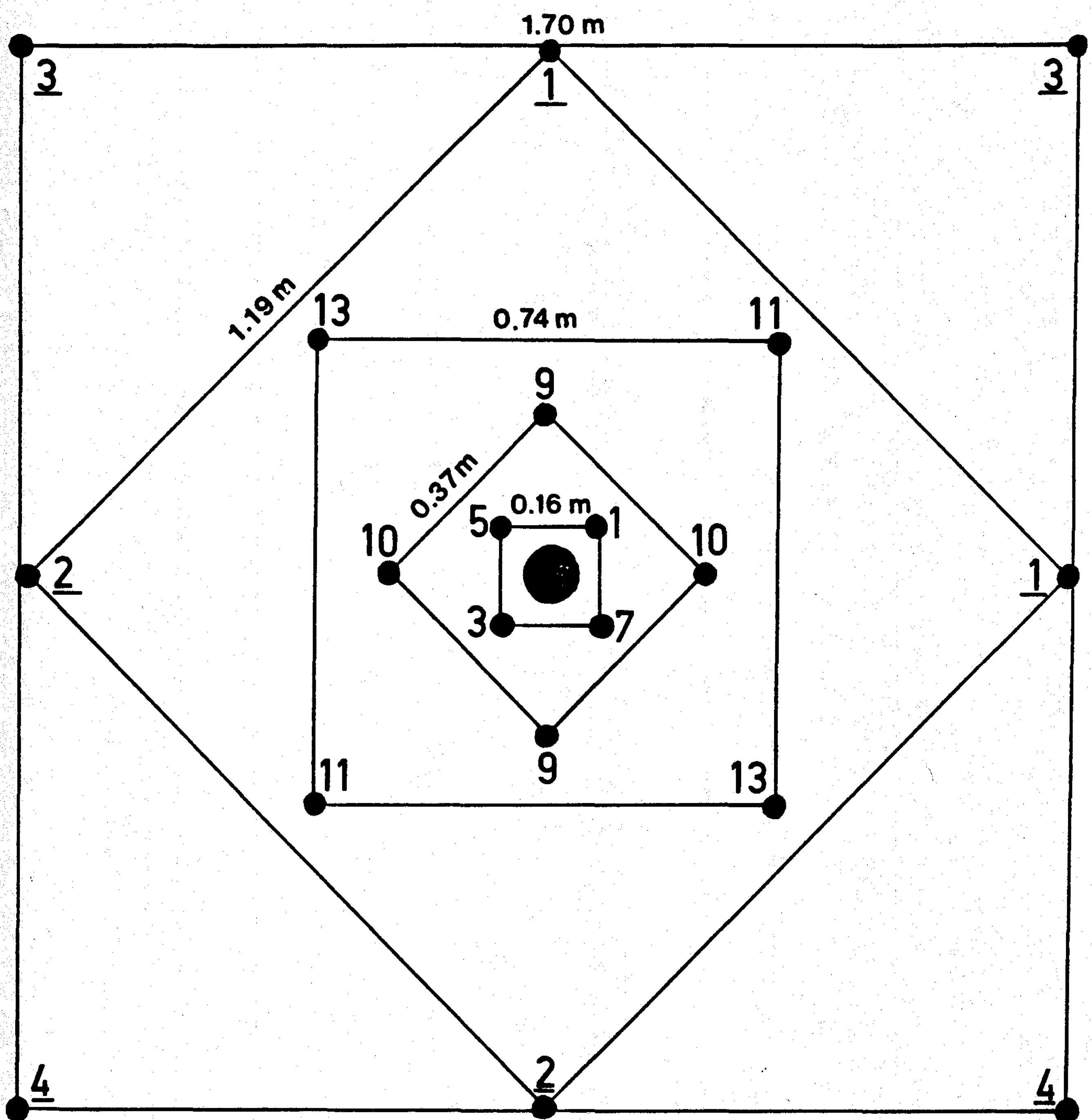
Seven different patterns were used as part of the Swedish blast design phase. The details of the explosive consumption and loading patterns for each of these rounds can be found in Appendix B. The details of hole locations and hole deviation can be found in Appendix C. In this section, only a summary discussion of each round will be presented.

4.3.2 Cut Design

The cut shown in Figure 4-1 was designed and used for rounds one through six. In round seven, a blast hole diameter of 45 mm and a hole length of 3 m required a modified cut. The cuts functioned very well, pulling (breaking) completely to the bottom of the holes.

4.3.3 Round One Design

The ignition pattern used for round one is shown in Figure 4-2. The "box" with the holes marked by asterisks corresponds to the cut shown in Figure 4-1. The numbers located beside the holes signify the Accudet delay number (Table 4-5). Right angle corners are avoided in order to prevent the generation of large radial cracks from highly constricted corners.



Underlined delay number denotes Accudet Mark V caps.
DuPont MS caps are used in the first three quadrangles.

Figure 4-1. The Designed Cut for Rounds 1-6

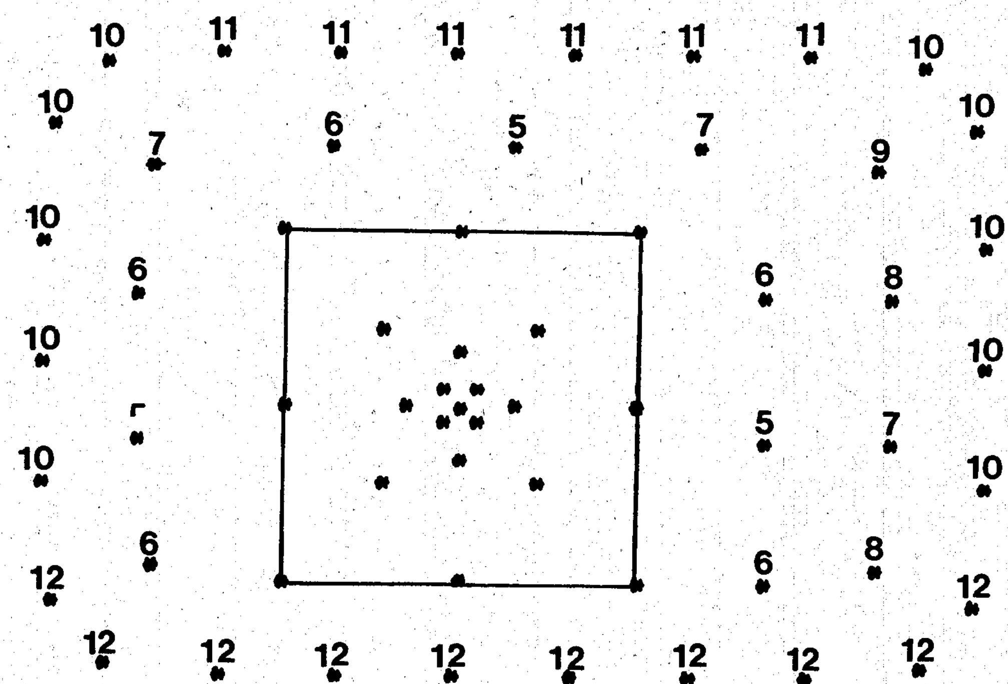


Figure 4-2. Ignition Pattern for Round 1

Tovex 210 and 220 were used as the explosives in the cut. Tovex 100 was used in the stoping holes next to the contour holes. In the contour holes, the charge concentration was only 0.12 kg/m and no tamping of the cartridges was done. This low charge density was achieved by using Ensign Bickford 200-grain seismic PETN-cord. Three 2.2-m lengths of PETN-cord were used in each of the contour holes (Figure 4-3). Each of the lifters, however, had to be loaded with four sticks of Tovex 100 and one stick of Tovex 210, since the hole deviation in this first round was too large for the planned lifter charge.

In this round, unsatisfactory breakage of the perimeter holes in the wall and the roof occurred. Only holes 52 and 53 (see Appendix B) had acceptable bootlegs. The fragmentation was coarse, but acceptable. A comparison of the average explosive consumption in the rounds blasted in the room prior to design blast round one and that of round one is given in Table 4-4. It is observed that with Design 1, the same volume of rock was removed using less than half the explosive previously used.

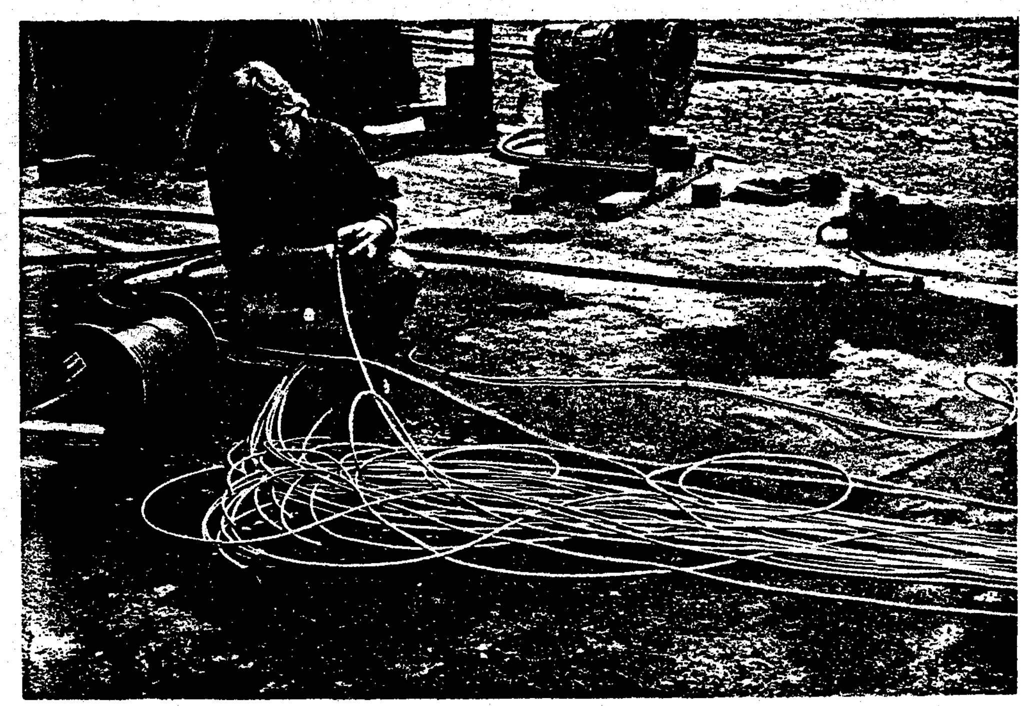


Figure 4-3. Preparation of PETN-Cord for the Contour Holes

Table 4-4. Comparison of Explosive Consumption Between Normal and Careful Blasting

Round	Explosive Consumption (kg/m^3)
Before Blast Round One	5.0
Blast Round One	2.2

4.3.4 Round Two Design

In round two, four holes were added to the pattern. The ignition pattern is shown in Figure 4-4. One hole was added in each perimeter row to allow reduced spacing in the contour. This was done since round one had an irregular contour, resulting from a large number of joint sets and hole deviations larger than expected. A smaller hole spacing would, of course, reduce this effect. The cut and the vertical stoping holes were lowered 5 cm to make it easier for the lifters to break. The lifters in this round were loaded with four 2 m lengths of 200-grain seismic PETN-cord (ℓ = 0.16 kg/m). Half a cartridge of Tovex 100 was used as a bottom charge for each of the contour holes with the exception of the lifters where one cartridge was used.

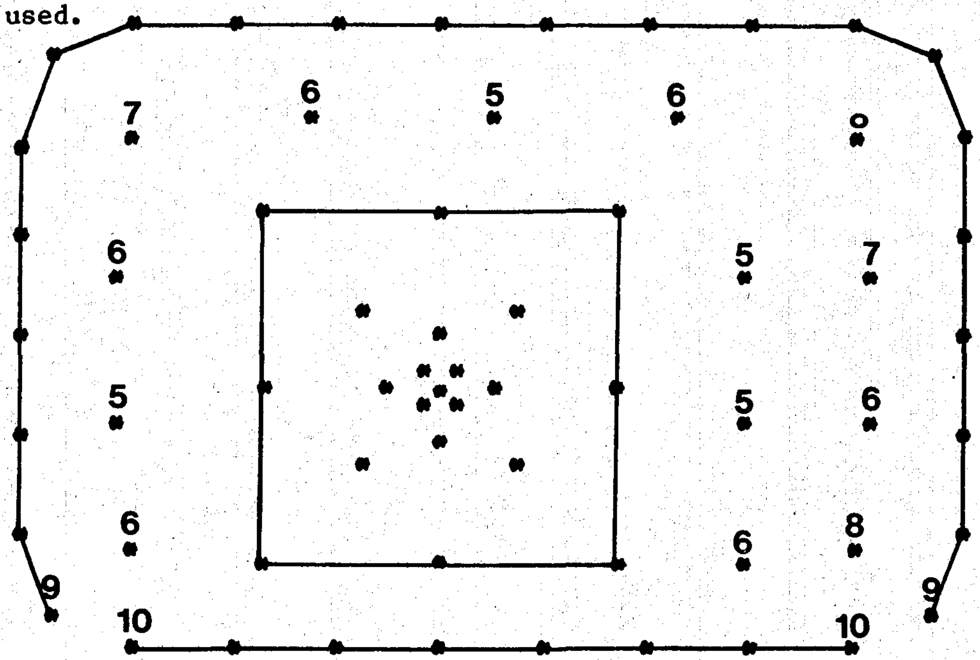


Figure 4-4. Ignition Pattern for Round 2

4.3.5 Round Three Design

It was decided to be more careful with the drilling of the lifters in round three, and as the blasting results for the rest of the round were good, the pattern for round three was not changed from that used in two. The only change was to ignite the lifters with an eight Accudet delay instead of a ten, in order to make it easier for the lifters (with such a low charge concentration) to heave the bottom. It was extremely difficult to load the broken rock with the Bobcat because the breakage at the floor level was not satisfactory. This was true even though the hole deviations for the lifters were almost acceptable. A lot of "teeth" were developed at the floor level due to joint intersections. If joints divide the spacing of the contour holes into several blocks, an acceptably smooth contour can be achieved when each block is intercepted by a contour hole. Therefore, the best remedy for these negative effects is to increase the number of holes in the contour and lower the charge weight per hole. This was done for the rest of the rounds.

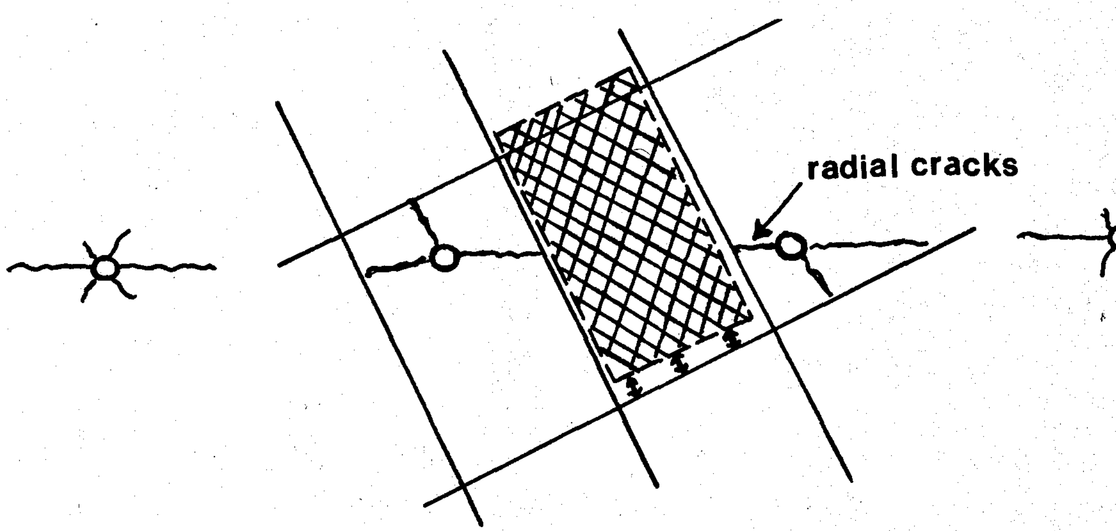


Figure 4-5. Schematic Drawing of Results Obtained When the Spacing of the Contour Holes is Large and When Intersecting Joint Sets Exist

Figure 4-5 illustrates what happened in rounds two and three. The distances between the existing intersecting joint sets were smaller than the hole spacing for the lifters. When detonation occurred, the length of the radial cracks was determined by the distance from the hole to the closest joint. The gas penetrated into the joints and lifted or loosened the intermediate block between two contour holes. The digging efficiency was lowered to a great extent, and the required hand mucking of the floor was very time consuming, hard work.

The depth to which the blocks broke could be as much as 0.4 m below the desired floor level.

As noted earlier, the experimental room had a bearing of S23E. Most of the problems with the floor were caused by the metamorphic foliation (N69E, 72NW) together with an almost perpendicular intersecting joint set (N54W, Vertical). Occasionally, two other joint sets (N7W, 30NE) and (N45W, 50SW) disturbed the planned contour.

4.3.6 Round Four Design

In round four, three more lifters were drilled and the charge per hole was reduced. The ignition pattern is shown in Figure 4-6. Half a cartridge of Tovex 100 was used as a bottom charge and only three 2-m long 200-grain seismic PETN-cords were used per hole. A number-nine cap was used for the lifters, but since the scatter for this delay is several hundred milliseconds (see Table 4-3), the holes were also connected with a 50-grain PETN-cord. It is a well known fact that improved smooth blasting results are achieved if the holes are fired simultaneously. If the PETN-cord was not cut by the previously fired holes in the round, the first number-nine cap in the lifters would initiate the PETN-cord and then all the lifters would be fired within 0.001 second. A simultaneous ignition also makes it easier to heave the bottom which increases mucking efficiency. The vibration measurements indicated that all the lifters were fired simultaneously.

This floor was the best one achieved to that date. At the left wall, the miners had problems when drilling the holes. Three drill rods had been stuck due to a joint, and hole 47 could not be drilled. After the blast, the left rib holes had to be reshot using PETN-cord. The right rib is shown in Figure 4-7.

4.3.7 Round Five Design

Round five had the same number of holes and the same specific charge as round four. The only difference was that the rib (side) and back (roof) holes were also connected with a PETN-cord. The ignition pattern is shown in Figure 4-8. The drilling accuracy for this round was higher than for all the other rounds. Vibration recording showed that the attempt to initiate the contour holes simultaneously did not work completely this time.

The lifters detonated in two shots with a time interval of 70 ms. For the rib holes, four shots were registered within 140 ms and the back holes were ignited in

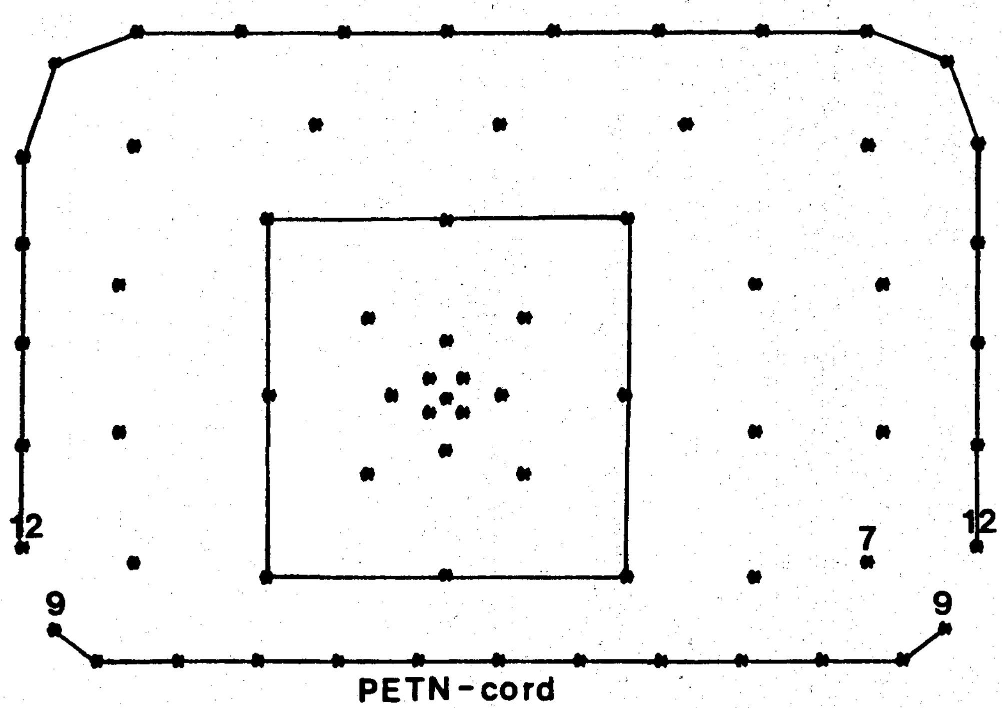


Figure 4-6. Ignition Pattern for Round Four

Figure 4-7. The Right Rib of the Blasted Round Four

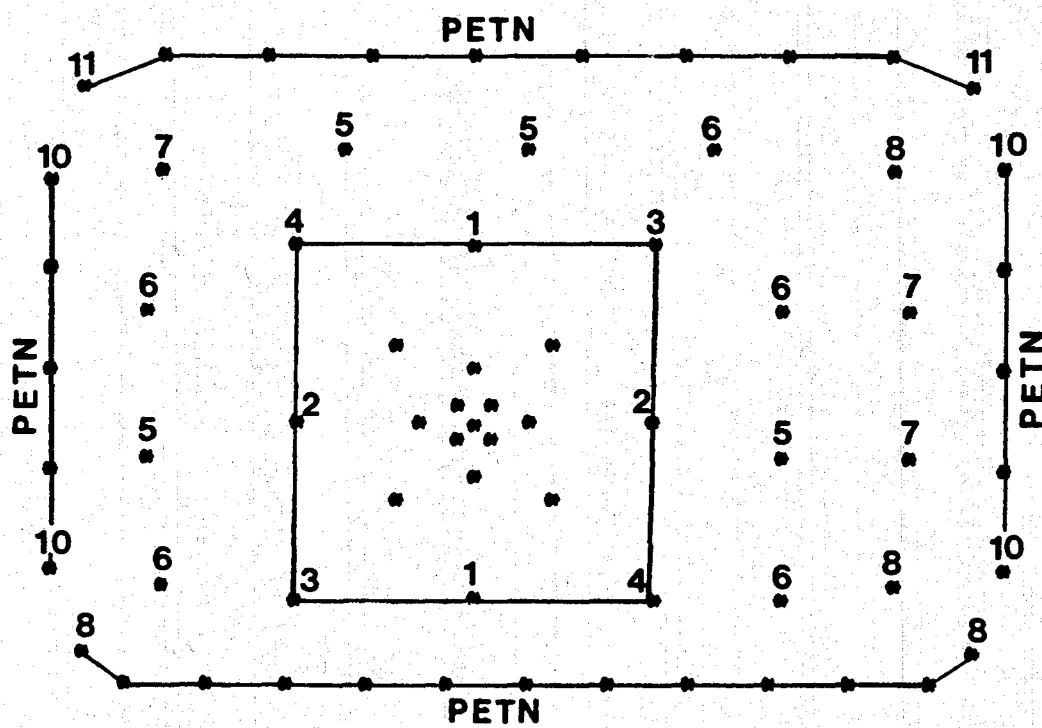


Figure 4-8. Ignition Pattern for Round Five

two shots separated by 140 ms. However, the result was very good and, in fact, was the best round shot (see Figures 4-9 and 4-10).

A vertical shear zone, having a thickness of 0.1 to 0.4 m, filled with clay and crushed material (some chloritized), crossed the round from left to right (see Figure 4-11). This shear influenced the blasting result such that some of the holes behind the shear had to be reshot.

4.3.8 Round Six Design

Round six was drilled with a 3-m hole depth and with a hole diameter of 45 mm. The ignition pattern is shown in Figure 4-12. A new drilling crew was employed and although they had worked for a contractor before, the drilling deviations were extremely large. The crew also made the mistake of connecting the caps in only one circuit. The result was a very poor round and a lot of redrilling and reshooting had to be done. DuPont's Tovex T-1 was used for contour blasting of the left half of the round and seismic cords were used for the right part. Although the round was a failure, it revealed that Tovex T-1 with a concentration of 0.37 kg/m (0.25 lbs/ft) definitely had too much strength for smooth blasting with the hole diameters used. The contour became considerably rougher with the Tovex T-1.

4.3.9 Round Seven Design

Round seven was redesigned to allow for the large hole deviations. The ignition pattern is shown in Figure 4-13. In the calculations, the deviation for the first three quadrangles were included as one degree. The rest of the holes were allowed to have a deviation of three degrees, and the lookout should be five degrees.

In the contour, the same hole distances were used for the lifters as had been used in round six. For the rib- and back-holes, the spacing was changed to be the same as for the lifters.

The holes next to the contour were designed to have the same lookout angle as the contour holes. With this, the contour holes and the stoping holes in the first row are parallel to each other. The specific charge for the contour holes (except the lifters) had been changed from 0.12 kg/m to 0.08 kg/m.

Except for the right upper corner, the round broke correctly. The bootlegs for holes 75, 76, 78, 79, 81, 83, 84, and 85 (see Appendix B) were larger than could be tolerated, and these were reshot with half a stick of Tovex 100. However, for some reason, holes 82 and 86 had not been drilled and this probably affected the result.

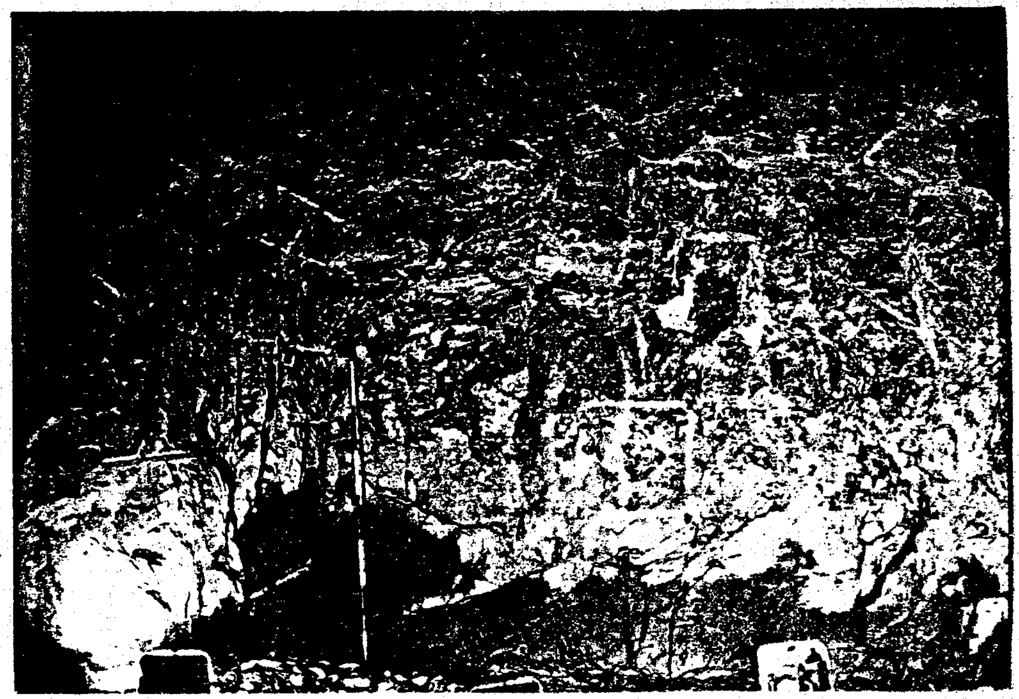


Figure 4-9. The Blasted Round Five and the Face of Round Six

Figure 4-10. The Smooth Solid Floor After Round Five

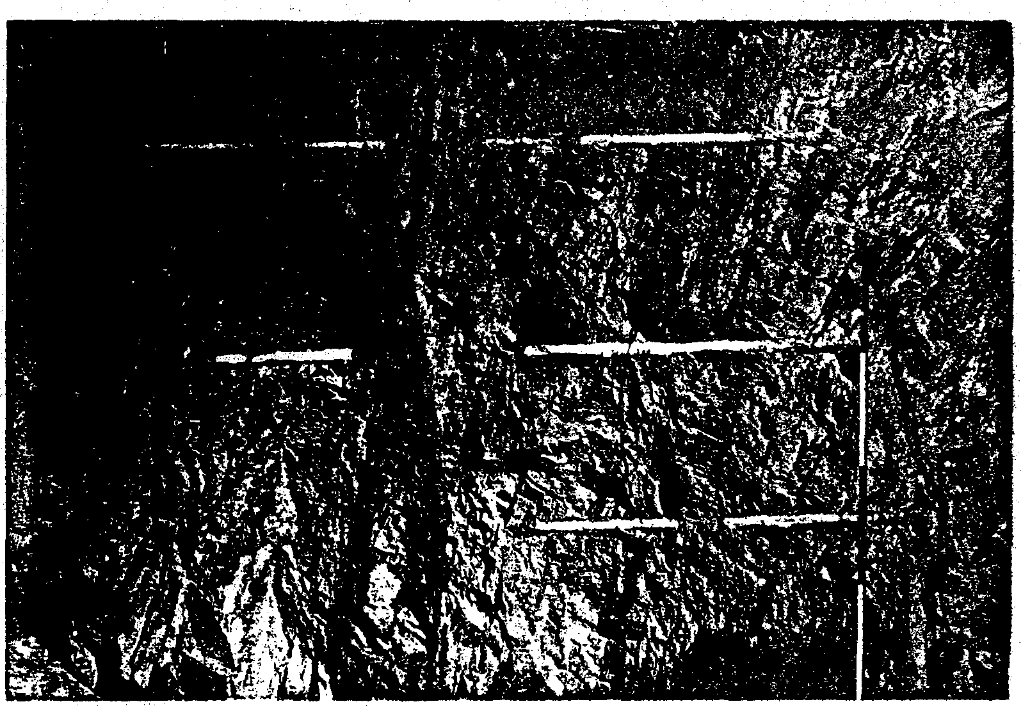


Figure 4-11. Vertical Shear on Left Rib in Round Five

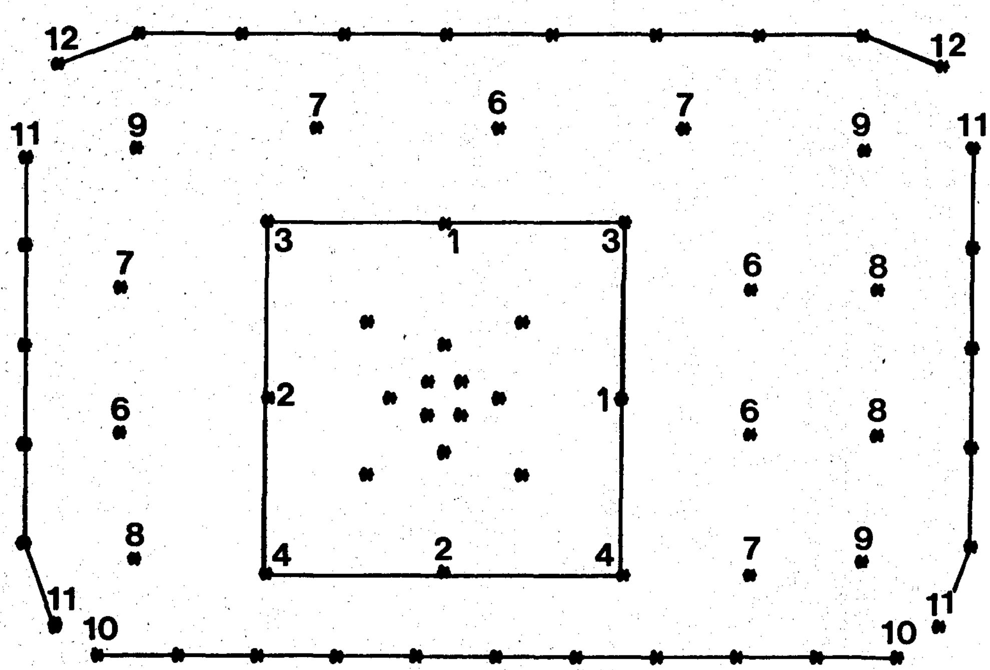


Figure 4-12. Ignition Pattern for Round Six

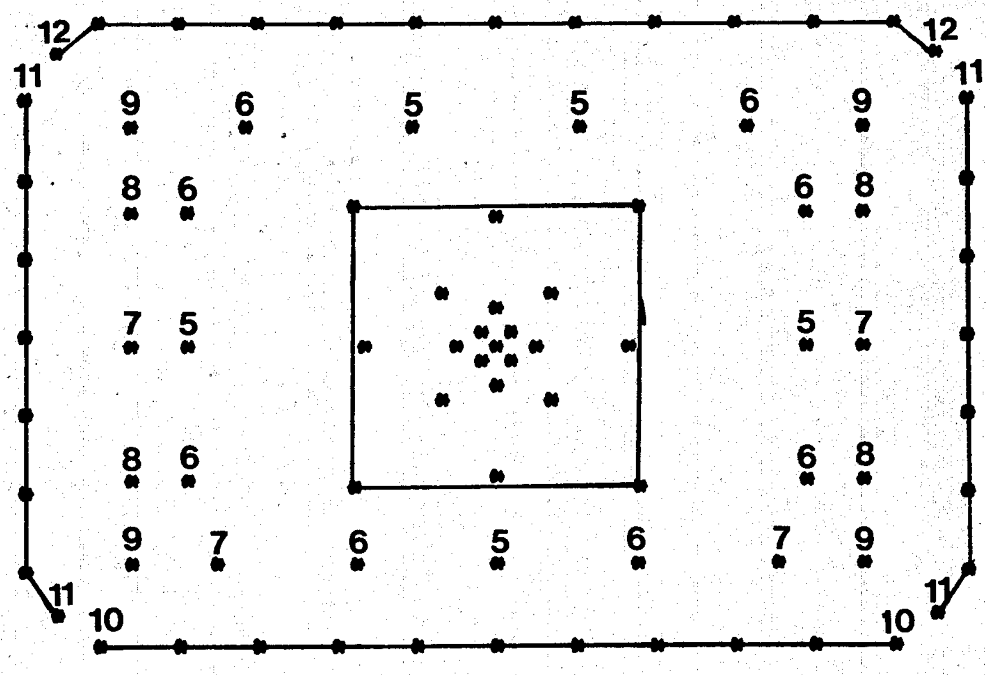


Figure 4-13. Ignition Pattern for Round Seven

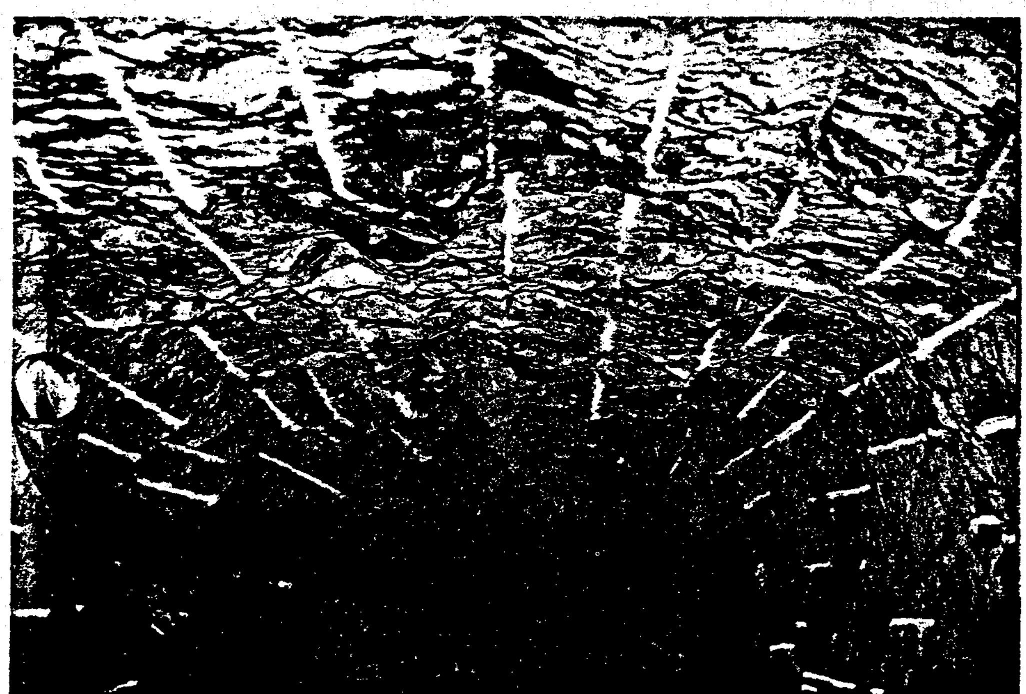


Figure 4-14. The Rib and Back of Rounds 1-5 Seen from the Face of Round Six Towards the Entrance

Table 4-5 summarizes the explosive consumption, specific drilling, and advance data for the seven rounds.

Table 4-5. Data for the Seven Rounds

Round	No. of Holes	Hole Depth (m)	Advance %	Tovex 100(kg)	Tovex 210(kg)	Tovex 220(kg)
1	61	2.22	85	17.6	20.5	22.4
2	65	2.28	15.7	20.5	24.0	8.0
3	65	2.19	92	16.9	23.0	15.3
4	68	2.25	92	12.9	28.2	16.0
5	68	2.23	97	13.1	28.2	15.8
6	68	3.0	15.6	33.9	20.8	5.5
7	86	2.4	94	25.3	12.2	36.8

Round	PETN-cord(kg)	Tovex T-1(kg)	Total Charge(kg)	Specific Charge(kg/m^3)	Specific Drilling(m/m^3)
1	5.3	-	65.8	2.20	4.52
2	8.0	-	68.2	2.22	4.81
3	8.2	-	63.4	2.14	4.81
4	8.1	-	65.2	2.15	5.04
5	7.9	-	65.0	2.16	5.03
6	5.5	18.2	94.0	2.32	5.04
7	7.5	-	81.8	2.52	6.37

Remarks: The bootlegs were not measured for rounds two and six. Round two had about the same advance as round three. Round six was a misfire and had to be reshot.

5 SURVEYING FOR HOLE DEVIATION

Each hole in every round (except number seven) was carefully surveyed to determine the horizontal and the vertical deviations. A compass was used to take the bearing direction of each hole. This could be done with a maximum error of ±0.5 degrees. With the bearing, the horizontal angular deviation (aligning deviation) could be easily evaluated. To get the vertical deviation, a stick was placed in the drilled hole, and a device for measuring the deviation from the horizontal was placed on the stick. The estimated error in the vertical deviation measurement is also ±0.5 degrees.

As the hole depth was also logged for each of the holes, the total deviation at the hole bottom (assuming that the hole did not curve) could be easily evaluated.

This check on deviations was done primarily for the following reasons:

1) to allow redrilling or heavier loading,
2) to check lookout angles, and
3) to obtain correlation between the length of the bootlegs (or the advance) and the deviations.

The preliminary designs of the rounds were specified to have a lookout angle of three degrees and an angular deviation of ±1.7 degrees. However, there was no way to achieve this with the available drilling equipment. Therefore, the specification was changed to five degrees for lookout angles with a tolerated deviation of three degrees. Since only 2.4-m long holes could be drilled, it was necessary for them to bottom out at least 20 cm outside the new round in order to assure enough space to collar the new contour holes.

If the drillers had been able to drill as designed, Figure 5-1 would represent the results of the hole surveying.

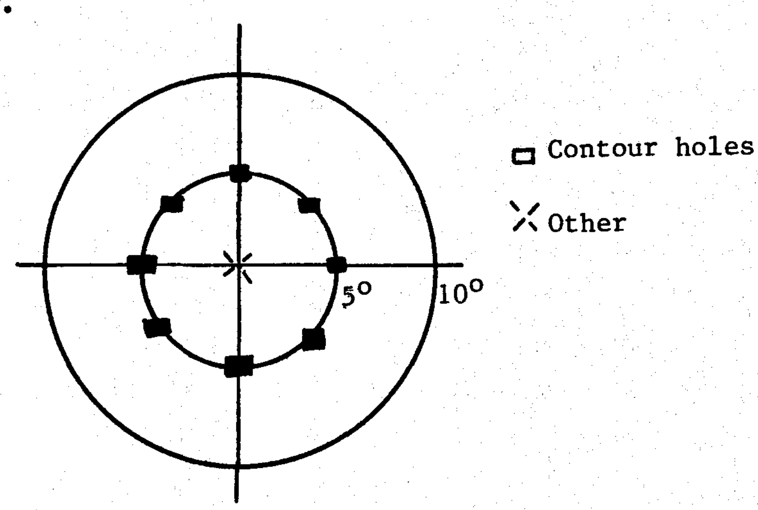


Figure 5-1. Theoretical Figure of the Hole Directions

All stoping holes should hit the zero point and the perimeter holes should be a long circle of five degrees (the lookout angle) at a spot depending upon whether it was a lifter, a corner hole, a ring hole, or a back hole.

The surveying data were fed into a computer and figures were plotted for all the holes in each round. Mean values and standard deviation for both vertical and horizontal hole deviations were calculated. Histograms and normal distribution curves were also calculated and plotted. To better visualize the effects of drilling deviation and to indicate the rock broken outside of the theoretical contour, figures were drawn on which the beginning (collar) and the end (toe) of each of the holes are marked. The detailed figures are shown in Appendix C.

Mean values and standard deviations for all holes except the contour holes are summarized in Table 5-1.

Table 5-1. Data from a Statistical Analysis of Angular Deviations for All Stoping and Cut Holes

Round No.	Vertical Angular Deviation (degrees)	Horizontal Angular Deviation (degrees)
1	1.94 ± 3.30	0.00 ± 5.29
2	-0.69 ± 1.64	1.94 ± 4.15
3	-1.63 ± 3.23	-1.37 ± 3.35
4	-0.57 ± 3.11	-0.89 ± 2.85
5	-0.29 ± 2.30	1.57 ± 3.07
6	-0.20 ± 3.27	-0.60 ± 3.97

Table 5-1 reveals that the standard deviations are much higher than those which should be tolerated to get a perfect result. The first round drilled was the worst one, but progress was made up to round five which was the best one. The data presented in Table 4-5 indicates that the advance increased from 85% to 97% of the drilled depth from round one to five. This progress definitely indicates the benefit of accurate drilling. The drilling crew that drilled rounds one to five were mining students at CSM with relatively little experience in drilling with the jumbo and the jacklegs. Rounds six and seven were drilled by a new crew of students that had some years of experience in drilling in a mine. From the drilling deviation figures, one can observe that the accuracy became lower, and so did the advance. This definitely indicates the importance of having an instrument for aligning the holes in a proper way. A jumbo with parallel guided booms and an automatic device for setting the lookout angle would have prevented some of this deviation. A large hole deviation not only results in a higher specific charge and specific drilling for the blasting operation, but it also affects the blasting result. As rock outside the theoretical contour is excavated, more concrete has to be used (if reinforcing is required), higher ground vibrations will be experienced, and sometimes more rock damage is produced.

6 GROUND VIBRATION MONITORING

The connection between ground vibrations and blasting damage to nearby facilities has long been known, and fairly reliable damage criteria have been established. The application of the same basic principles for assessing or predicting damage to the contour exposed by the blast itself is now being attempted.

A blast monitoring program using a SINCO Model S-3 vibration monitor was carried out simultaneously with each round. Initially, it was planned to make all the vibration measurements in A-left spur (which is almost parallel to the experimental room), thus maintaining a constant distance of about 30 m for each round. The measurement of each particle velocity would provide an idea of how the different blasting patterns affected the rock mass.

The frequency response of the selected vibration equipment was considerably lower than that of the wave motion at the site. The SINCO Model S-3 had a frequency response from 6 to 150 cycles per second. The frequency range measured at the site was between 200 to 600 cycles per second.

As a result, after round seven, the monitoring gages were moved into Miami drift about 100 m from the experimental room. This reduced the accuracy in predicting the rock damage. The data accumulated was thus mainly used to check the ignition times for different charges.

Two gages, each containing three transducers (one for each of the longitudinal, vertical, and transversal velocity components) were used for making the measurements. One gage was mounted on the back of the drift and one on the rib closest to the blast. An evaluation of the measurements revealed no significant differences between velocities measured on the back or rib. The data showing average peak particle velocities and frequency for wave motion for each round are presented in Tables 6-1 and 6-2.

The following equation could be used to predict peak particle velocities in the range of 30 to 100 m away from the rounds.

$$V = \frac{730}{\left(\frac{R}{Q^{0.43}}\right)^{1.54}} \tag{6-1}$$

where R denotes the distance in meters, Q denotes the charge weight in kg and v is the predicted particle velocity in mm/s.

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Table 6-1. Results from Vibration Monitoring Giving Average Peak Particle Velocities and Average Frequency

Round No.	Peak Particle Velocity Burn Cut (in/sec)	Peak Particle Velocity Remaining Holes (in/sec)	Average Frequency (cycles/sec)
1*	0.316	0.239	337.3
2*	0.409	0.316	425.0
3	0.035	0.024	214.7
4	0.080	0.009	240.0
5	0.350	0.289	909.0
6	0.038	0.033	241.5
7	0.035	0.053	210.9
8	0.027	0.032	244.7
9	0.047	0.032	230.8
10	0.054	0.063	219.7

* Measured from A-left spur.

Table 6-2. Velocity Measurements for Delays on Round Two

Delay Number	Delay Time (sec)	Number of Holes	Velocity in/sec (rib) L	V	T	VT
MS	0.025-0.450	12 Burn Cut	0.272	0.2025	0.228	0.409
1 (ACUDET)	0.5	2	0.232	0.130	0.163	0.312
2 "	1.0	2	0.165	0.200	0.246	0.357
3 "	1.5	2	0.269	0.139	0.216	0.372
4 "	2.2	2	0.269	0.139	0.216	0.372
5 "	3.0	4	0.227	0.134	0.217	0.341
6 "	3.8	6	0.217	0.206	0.216	0.369
7 "	4.6	2	0.252	0.212	0.210	0.393
8 "	5.5	2	0.236	0.086	0.223	0.336
9 "	6.4	22 Perimeter	0.113	0.125	0.080	0.188
10 "	7.4	8 Holes	0.131	0.100	0.060	0.175

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APPENDIX A
CHARGE CALCULATIONS FOR TUNNELING

INTRODUCTION

The driving of drifts is a very important aspect of underground excavation. In this paper, empirical relationships that can be used for designing tunnel and drift blast rounds will be presented. The basic principles of the calculation method are based upon the earlier work of Langefors-Kihlstrom [(1)] and Gustafsson [(2)].

COMPARISON OF EXPLOSIVES

To provide for the use of various explosives, it is necessary to have a basis of comparison. Several methods have been developed to characterize the strength of an explosive. The basis for comparison is: 1) comparison of energies determined using the ballistic mortar, the Trauzl lead block test, or the underwater test; 2) brisance; 3) grade strength; or 4) weight strength. However, most of these should be used carefully when stating the breaking capacity of an explosive in a rock material. For example, depending upon the type of the blasting operation; i.e., crater blasting, bench blasting, etc., the strength of the explosive must be estimated from different premises.

The best way to rank explosives would be of course to measure the rock breakage capacity in different rock materials with different blasting operations under different charging conditions. Such an evaluation is, however, prohibitive due to the costs and time involved. Instead, one usually is restricted to using one of the aforementioned methods for the comparison of the strength.

In this paper, the Swedish weight strength relationship is used for the correlation of different explosives. The relation is described by

$$s = \frac{5}{6}\frac{Q}{Q_o} + \frac{1}{6}\frac{V}{V_o} \qquad \text{(A-1)}$$

where

s = weight strength relative to a reference explosive (LFB-dynamite)

Q_o = heat of explosion for 1 kg of LFB

V_o = released gas/volume from 1 kg of LFB (at STP)

Q = heat of explosion for 1 kg of the actual explosive

V = released gas/volume from 1 kg of the actual explosive

Q_o = 5.0 MJ/kg, V_o = 0.85 m^3/kg .

The formula is based upon the fact that the work of expansion depends primarily upon the heat of explosion and secondarily upon the released gaseous reaction products. The constants 5/6 and 1/6 in the formula were determined in field experiments where los and high gas volume explosives were used and compared to LFB-dynamite under bench blasting conditions. Today, the weight strength is seldom expressed relative to LFB. Usually it is given with respect to either ANFO, or as in Nitro Nobel AB's product prospective, relative to the dynamite Dynamex B. When weight strength is expressed relative to ANFO, one must first calculate the weight strength relative to LFB and then divide the value by the weight strength of ANFO relative to LFB (0.84).

Table A-1. Weight Strength for Some Explosives

Explosive	Q MJ/kg	V m^3/kg	s_{LFB}	s_{DXB}	s_{ANFO}	Density kg/m^3
LFB Dynamite	5.00	0.850	1.00	1.09	1.19	
Dynamex B	4.6	0.765	0.92	1.00	1.10	1450
ANFO	3.92	0.973	0.84	0.91	1.00	900
TNT	4.1	0.690	0.82	0.89	0.98	1500
PETN	6.12	0.780	1.17	1.27	1.39	
Nabit	4.1	0.892	0.86	0.93	1.02	1000
Gurit	3.73	0.425	0.71	0.77	0.85	1000

Generally, the weight strength concept better describes the magnitude of the expansion work that the blasting agent can perform in a blasting operation than does the released energy along. One must keep in mind, however, that it is impossible to utilize the total energy for breaking rock. The explosion energy is the released chemical energy. To utilize all of this energy as expansion work, the gaseous products must have the possibility of expanding to a very low pressure. Rock breakage and the primary fragmentation is already completed when the detonation products have expanded to a volume of about ten times the initial borehole volume. The pressure in the products at this expansion is in the range of 10-100 MPa.

Depending upon the ingredients in the explosive, especially the solid ones, the efficiency can vary considerably. Aluminized explosives, for example, obviously have a high total explosion energy. Unfortunately, a high proportion of their expansion work occurs in the low pressure region which lowers the efficiency significantly (Figure A-1).

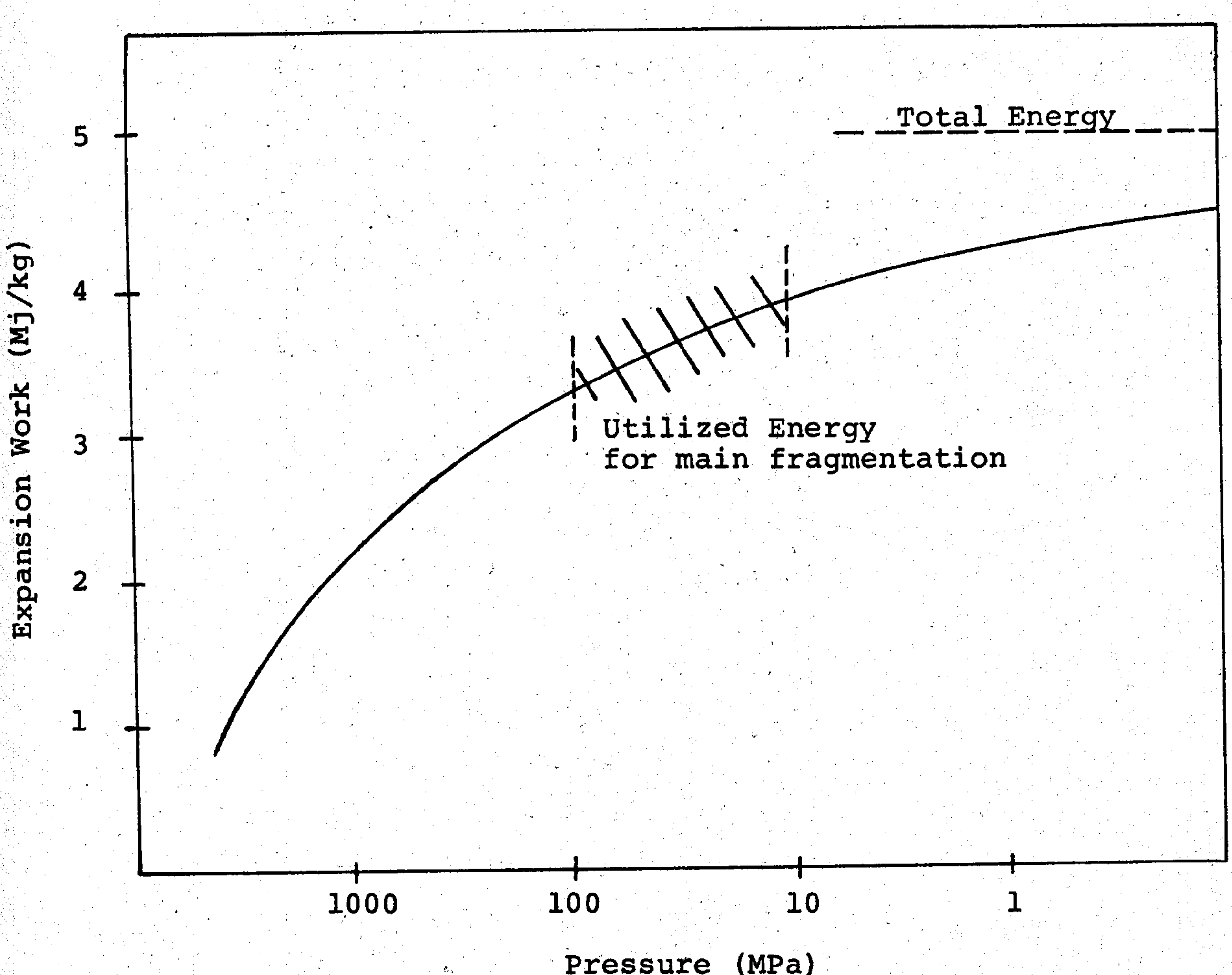


Figure A-1. Example of the Expansion Work as a Function of the Pressure in the Reaction Products for an Aluminized Watergel Explosive

CHARGE CALCULATION AND DESIGN OF DRILLING PATTERN

Tunnel blasting is a much more complicated operation than bench blasting because the only available free surface toward which initial breakage can occur is the tunnel face. Because of the high constriction, there will be a need for a much higher specific charge. Figure A-2 presents a good guide of explosive consumption for varying tunnel sizes.

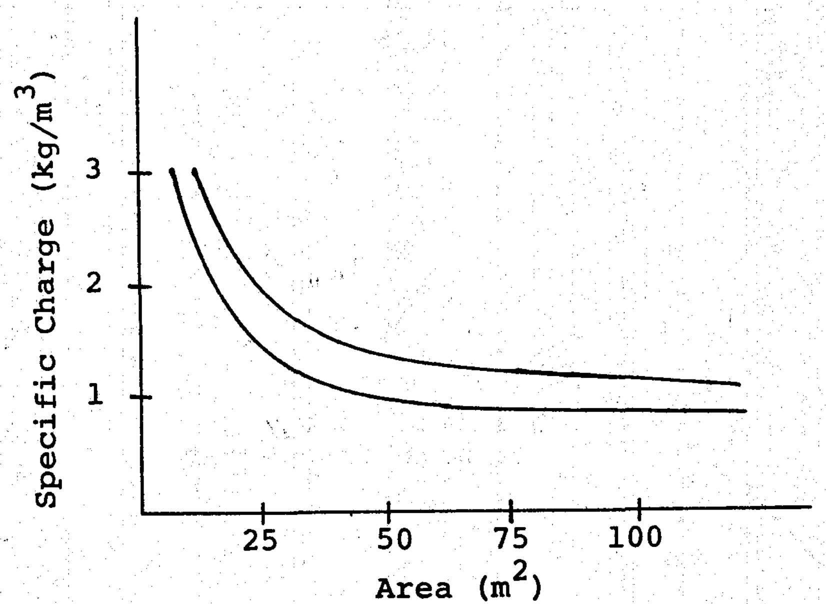


Figure A-2. Specific Charge as a Function of the Tunnel Area

Environmental aspects influence the choice of explosive by the avoidance of high concentrations of toxic fumes. The small burdens used in the cut demand an explosive agent which is sufficiently insensitive so that flashover from hole to hole is impossible, and has a sufficiently high detonation velocity to prevent the occurrence of channel effects when the coupling ratio is less than one. With the mechanized drilling equipment used today, holes larger than the required charge diameter are often drilled. Channel effects can occur if an air space is present between the charge and the borehole wall. If the detonation velocity is not high enough (less than about 3,000 m/s), the expanding detonation gases drive forward the air in the channel as a compressed layer with a high temperature and a high pressure. The shock front in the air compresses the explosive in front of the detonation front, destroys the hot spots or increases the density to such a degree that the detonation could stop or result in a low energy release. The explosive used in the lifters must also withstand water. In the contour holes, special column charges should be used to minimize damage to the remaining rock.

To simplify the charge calculations, let us divide the tunnel face into five separate sections A-E (Figure A-3). Each one has to be treated in its own special way during the calculation.

A = the cut section

B = the stoping holes breaking horizontally and upwards

C = stoping holes breaking downwards

D = contour holes

E = lifters

The most important operation in the blasting procedure is to create an opening (cut) in the rock face to serve as a second free surface. If this stage fails, the round will definitely not be a success.

In the cut, the holes are arranged in such a way that the delay sequence permits the opening to gradually increase in size until the stoping holes can take over. The holes can be drilled in a series of wedges (V-cut), as a fan, or in a parallel geometry usually centered around an empty hole.

The choice of the cut has to be done with respect to the type of available drilling equipment, the tunnel width and the desired advance. With V-cuts and fan cuts (where angled holes are drilled) the advance is strictly dependent upon the width of the tunnel. In the last decade, the parallel cut (four section cut) with one or two centered large empty diameter holes has been used to a very large extent. The obvious advantages to using this cut are that no attention has to be paid to the tunnel width and the cut is much easier to drill, as there is no need to change the angle of the boom.

The principle behind a parallel cut is that small diameter holes are drilled with great precision around a larger hole ($\emptyset$ = 65 to 175 mm). The larger empty hole serves as a free face for the smaller holes and the opening is enlarged gradually until the stoping holes take over.

The predominant type of parallel hole cut is the four section cut. This will be used in the following calculation.

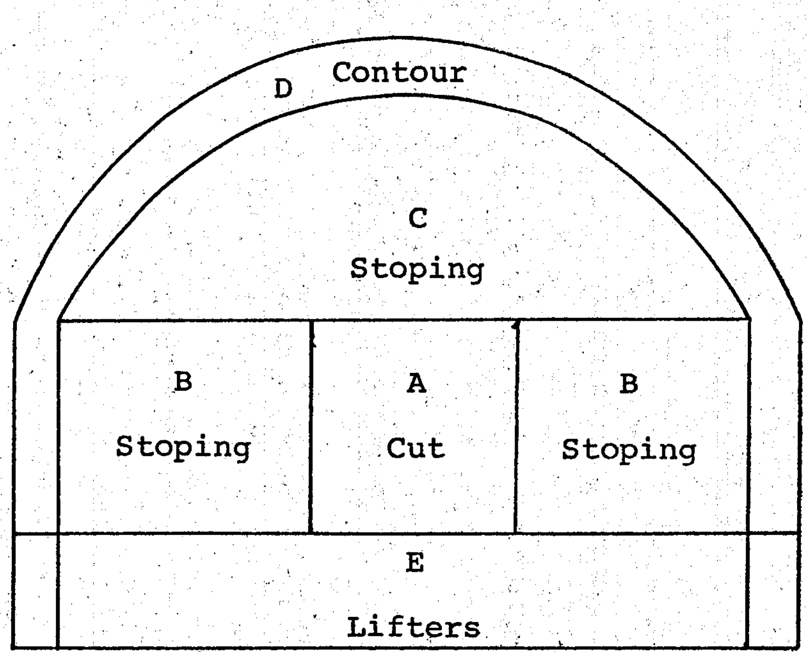


Figure A-3. Sections A-E Represent the Types of Holes Used Under Different Blasting Conditions

Advance

The advance is restricted by the diameter of the empty hole and the hole deviation of the smaller diameter holes. Good economics demands maximum utilization of the full hole depth. Drifting is very expensive if the advance becomes much less than 95 percent of the hole depths. Figure A-4 illustrates the required hole depth as a function of the empty hole diameter when a 95 percent advance is desired with a four section cut.

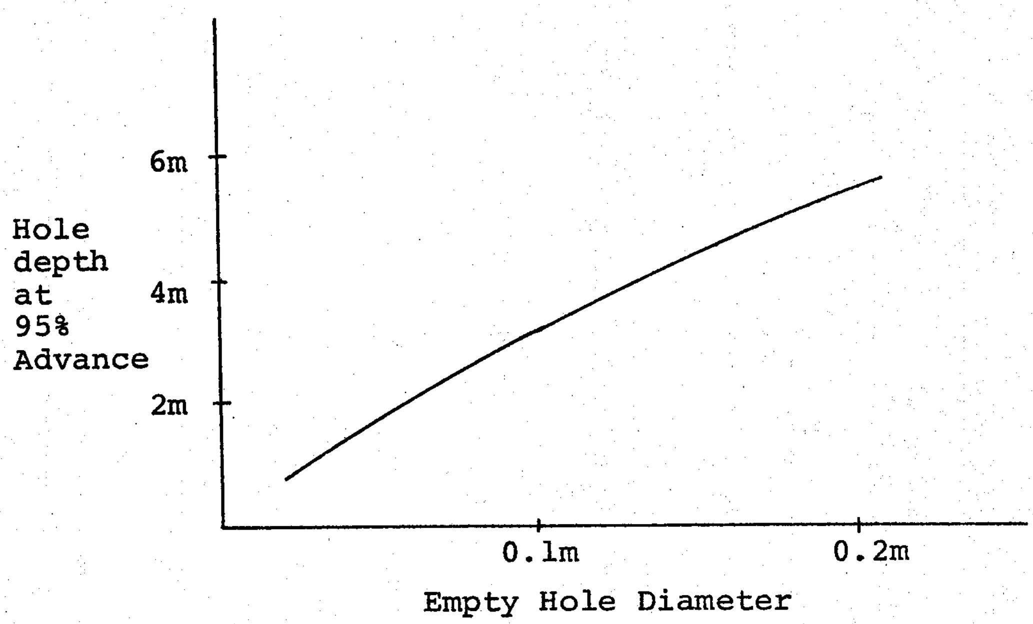


Figure A-4. Hole Depth as a Function of Empty Hole Diameter for a Four Section Cut

The equation for hole depth, H, can be expressed as

$$H = 0.15 + 34.1\ \emptyset - 39.4\ \emptyset^2 \qquad (m) \qquad (A-2)$$

where $\emptyset$ is the hole diameter in meters.

The advance I is

$$I = 0.95\ H \qquad (m) \qquad (A-3)$$

Equations A-2 and A-3 are only valid for a drilling deviation not exceeding 2 percent.

Sometimes two empty holes are used in the cut instead of one. This occurs, for example, if the drilling equipment cannot handle a larger diameter. Equation A-2 is still valid if $\emptyset$ is computed according to the following.

$$\emptyset = d_o \sqrt{2} \qquad (m) \qquad (A-4)$$

Here, d_o denotes the hole diameter of each of the two empty holes.

The general geometry for the cut and cut spreader holes is outlined in Figure A-5.

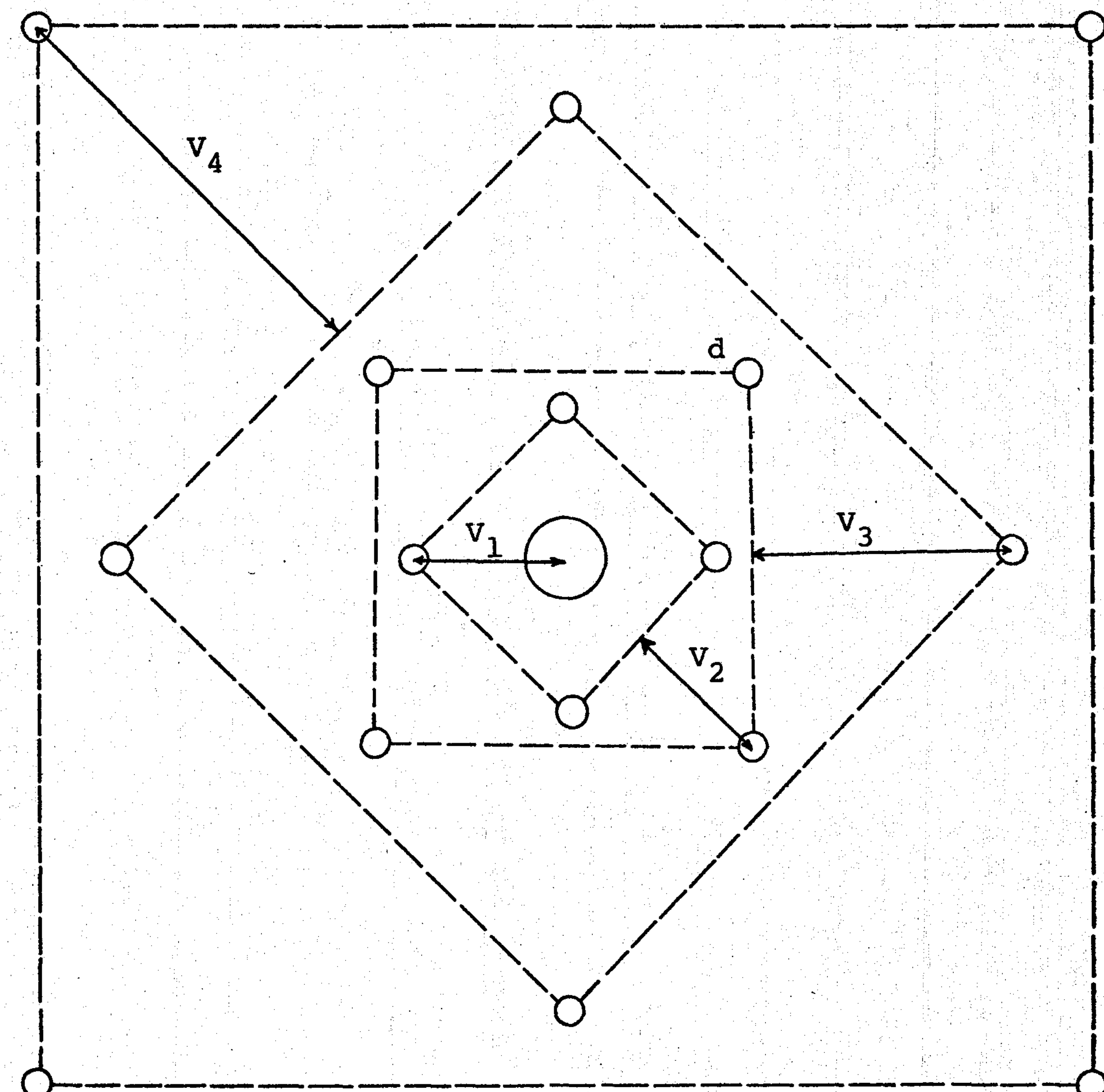


Figure A-5. Four Section Cut: V_1 Represents the Practical Burden for Quadrangle 1

Burden in the First Quadrangle

The distance between the empty hole and the drillholes in the first quadrangle should not exceed 1.7 times the diameter of the empty hole if satisfactory breakage and cleaninng are to take place. Breakage conditions differ very much depending upon explosive type, structure of the rock and distance between the charged hole and the empty hole.

As one can see in Figure A-6, there is no advantage in using a burden greater than 2$\emptyset$, as long as the aperture is to small for the heavy charge. Plastic deformation would be the only effect of the blast. Even if the distance is smaller than 2$\emptyset$, too great a charge concentration could cause a malfunction of the cut due to rock impact and sintering, which prevents the necessary swell. If the maximum accepted hole deviation is of the order of 0.5-1 percent, then the practical burden, V_1, for the spreader holes in the cut must be less than the maximum burden [V = 1.70]. We use

$$V_1 = 1.5\ \emptyset \qquad (m) \qquad (A\text{-}5)$$

When the deviation exceeds 1%, V_1 has to be reduced even further. The following formula should then be used.

$$V_1 = 1.7\ \emptyset - (\alpha\ H + \beta) \qquad (m) \qquad (A\text{-}6)$$

where the last term represents the maximum drill deviation, F, and

α = the angular deviation, (m/m)

H = the hole depth (m) and

β = the collaring deviation (m)

In practice, drilling precision is normally good enough to allow the use of equation A-5.

Charge Concentration in the First Quadrangle

Langefors and Kihlstrom (1) have verified the following relationship between charge concentration, ℓ, the maximum distance between the holes, V, and the diameter of the empty hole, $\emptyset$, for a borehole with a diameter of 0.032 m.

$$\ell = 1.5\ (V/\emptyset)^{1.5}(V - \emptyset/2) \qquad (kg/m) \qquad (A\text{-}7)$$

To utilize the explosive in the best manner, a burden of $V_1 = 1.5\ \emptyset$ (deviation of 0.5-1 percent) should be used.

One must remember that formula (A-7) is only valid for a drill hole diameter of 0.032 m. If larger holes are going to be used in the round an increased charge concentration per meter of borehole has to be used. To keep the breakage at approximately the same level, it is necessary to increase the concentration in

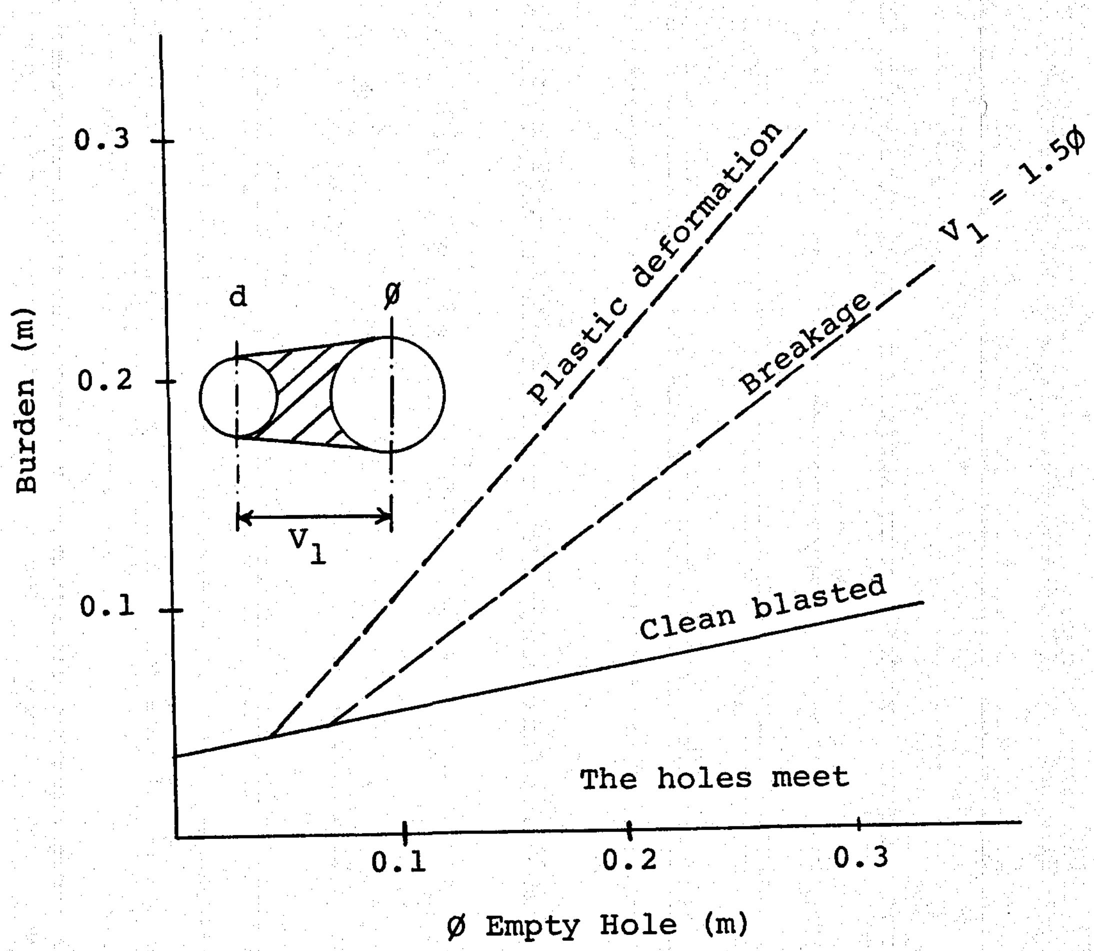


Figure A-6. Blasting Result for Different Relations Between the Practical Burden and the Empty Hole Diameter. Adapted from: Langefors and Kihlstrom(1) proportion to the diameter. Thus, if a drill hole diameter of d is used instead of d_1 = 0.032 m, the charge concentration is determined by

$$\ell = \frac{d}{d_1} \ell_1 \qquad \text{(A-8)}$$

Obviously, when the diameter is increased, this means that the coupling ratio and the borehole pressure decreases. It is important to carefully select the proper explosive in order to minimize the risk of channel effects and incomplete detonation.

Considering the rock material and type of explosive, equation (A-7) can now be rewritten in terms of a general hole diameter d:

$$\ell = 55\ d\ (V/\emptyset)^{1.5}\ (V-\emptyset/2)\ (c/0.4)\ /\ s_{ANFO}\ \text{(kg/m)} \qquad \text{(A-9)}$$

s_{ANFO} denotes the weight strength relative to ANFO and c is defined as the rock constant.

Often the possible values for charge concentration is given and the burden is calculated from formula (A-7) instead. This can easily be done using a pocket calculator.

Rock Constant

The factor, c, called the rock constant, is an empirical measure of the amount of explosive required to loosen one cubic meter of rock. The field experiments, by which the c values were determined, took place with a bench blasting geometry. It turns out that the rock constant determined in this way also gives a good approximation for the rock properties in tunneling. In trial blasting, it was found that c fluctuated very little. Blasting in brittle crystalline granite gave a 'c' factor equal to 0.2. In practically all other rock materials, from sandstone to a more homogeneous granite, a c value of 0.3 - 0.4 kg/m^3 was found. Under Swedish conditions, c = 0.4 is predominant in blasting operations.

The Second Quadrangle

After the first quadrangle has been calculated, a new geometry applies when solving for the burdens in the subsequent quadrangles. Blasting towards a circular hole naturally demands a higher charge concentration than blasting towards a straight face due to a higher constriction and a less effective stress wave reflection.

If (Figure A-7) there is a rectangular opening of width, B, and the burden, V, is known, the charge concentration, ℓ, relative to ANFO is given by

$$\ell = \frac{32.3\ d\ c\ V}{s_{ANFO}\{\sin(\text{atn}B/2V)\}^{1.5}} \qquad \text{(kg/m)} \qquad \text{(A-10)}$$

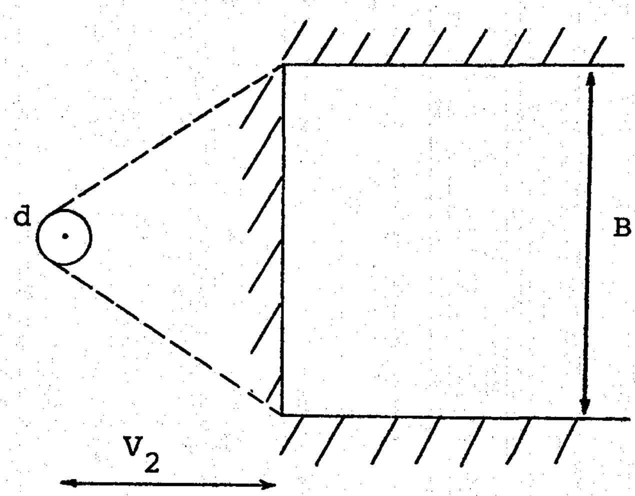


Figure A-7. Geometry for Blasting Towards a Straight Face

If instead, we start from the assumption that the charge concentration for the actual explosive and the rectangular opening width, B, are known, then the burden, V, can be expressed as a function of B and ℓ.

$$V = 8.8\times10^{-2} \quad \frac{B\ell \ s_{ANFO}}{dc} \qquad (m) \qquad (A\text{-}11)$$

When calculating the burden for the new quadrangle, the effect of faulty drilling, F (defined in equation A-6), must be included. This is done by treating the holes in the first quadrangle as if they were placed at the most unfavorable location.

From Figure A-8, one can see that the free surface, B, used in equation A-11, differs from the hole distance, B', in the first quadrangle.

$$B = \sqrt{2}\ (V_1 - F) \qquad (m) \qquad (A\text{-}12)$$

By substitution, the burden for the new quadrant is

$$V = 10.5\times10^{-2} \quad \left(\frac{(V_1-F) \qquad s_{ANFO}}{dc}\right)^{1/2} \qquad (A\text{-}13)$$

Of course, this value has to be reduced by the drill hole deviation to obtain the practical burden.

$$V_2 = V_2{-F} \qquad (A\text{-}14)$$

There are a few restrictions that must be put on V_2. It must satisfy the following

$$V_2 < 2B \qquad (A\text{-}15)$$

if plastic deformation is not to occur.

If it does not, then using equations (A-10) and (A-15), the charge concentration should be reduced to

$$\ell = \frac{32.3 \ dc \ 2B}{s_{ANFO}\{\sin(\text{atn } 1/4)\}^{1.5}} \qquad (kg/m) \qquad (A\text{-}16)$$

or

$$\ell = 540 \ d \ c \ B \ / \ s_{ANFO} \ . \qquad (A\text{-}17)$$

If the restriction for plastic deformation cannot be satisfied, it is usually better to choose an explosive with a lower weight strength in order to optimize the breakage.

The aperture angle should also be less than 90°. If not, then the cut will lose the character of a four section cut. This means

$$V_2 > 0.5 \ B \qquad (A\text{-}18)$$

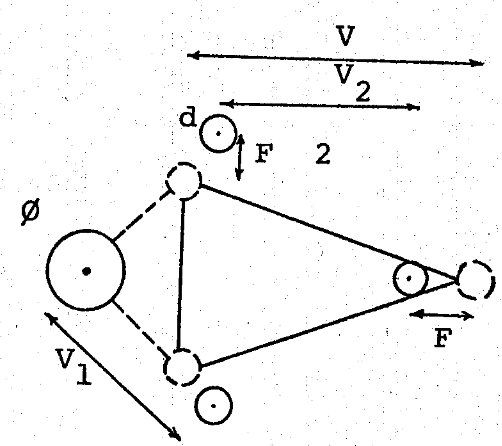


Figure A-8. Influence of the Faulty Drilling

Gustafsson [2] suggests that the burden for each quadrangle be $V_2 = 0.7\ B'$.

A rule of thumb for the number of quadrangles in the cut is that the side length (B') of the last quadrangle should not be less than the square root of the advance. The algorithm for the calculation of the remaining quadrangles is the same as for the second quadrangle.

Holes in the quadrangles should be loaded so that a hole length, h, of ten times the hole diameter is left unloaded.

$$h = 10\ d \tag{A-19}$$

Lifters

The burden for the lifters in a round are in principle calculated with the same formula as for bench blasting. The bench height is simply replaced by the advance and a higher fixation factor is used due to the gravitational effect and to a greater time interval between the holes. The maximum burden can be found using

$$V = 0.9 \left(\frac{\ell\ s_{ANFO}}{\bar{c}\ f\ (E/V)}\right)^{1/2} \quad (m) \tag{A-20}$$

where

f is the fixation factor

E/V denotes the relation between the spacing, E, and the burden, V

$\bar{c}$ is the corrected rock constant

$$\begin{aligned} \bar{c} &= c + 0.05, \text{ if} \quad V > 1.4 \text{ m, or} \\ \bar{c} &= c + 0.07/V, \text{ if} \quad V < 1.4 \text{ m} \end{aligned} \tag{A-21}$$

f = 1.45 and E/V = 1 is used for lifters.

When locating the lifters, one must remember to consider the lookout angle, γ (see Figure A-9). The magnitude of the angle is dependent upon the available drilling equipment and the hole depth. For an advance of about 3 m, a lookout angle equal to three degrees (corresponding to 5 cm/m) should be enough to provide room for drilling the next round.

Hole spacing should be equal to V. However, it will vary depending upon tunnel width.

The number of lifters, N, is given by

$$N = \text{integer of } \left(\frac{\text{Tunnel width} + 2\ H \sin\gamma}{V} + 2\right) . \tag{A-22}$$

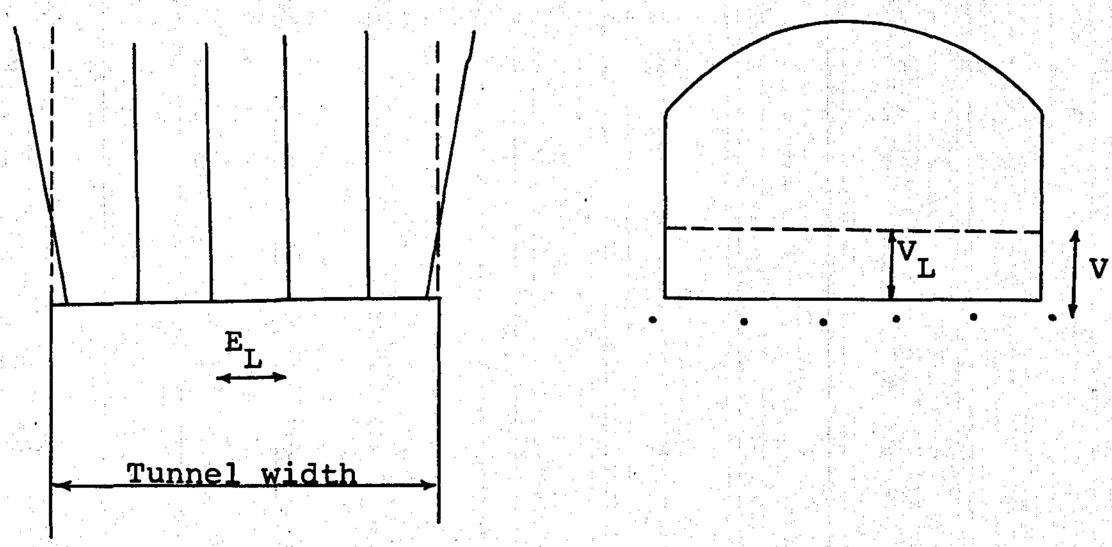


Figure A-9. Blasting Geometry for Lifters

The spacing E_L for the holes (with the exception of the corner holes) is evaluated by

$$E_L = \frac{\text{Tunnel width} + 2H \sin \gamma}{N-1} \quad . \qquad (m) \qquad (A-23)$$

The practical spacing E_L' for the corner holes is equal to

$$E_L' = E_L' - H \sin \gamma \, . \qquad (m) \qquad (A-24)$$

The practical burden V_L should be reduced by the bottom lookout angle and the faulty drilling.

$$V_L = V - H \sin \gamma - F \qquad (m) \qquad (A-25)$$

The length of the bottom charge, h_b, needed for loosening the toe is

$$h_b = 1.25\ V_L \, . \qquad (m) \qquad (A-26)$$

The length of the column charge, h_c, is given by

$$h_c = H - h_b - 10\ d \qquad (m) \qquad (A-27)$$

and the concentration of this charge can be reduced to 70 percent of the concentration in the bottom charge. However, this is not always done, since both it is time consuming. Generally, the same concentration is used in both the bottom and in the column.

For lifters, an unloaded length of 10 d is usually left at the collar. If equation (A-19) is going to be used, the following condition has to be fulfilled.

$$V < 0.6\ H \qquad (A-28)$$

Otherwise, the maximum burden has to be successively reduced by lowering the charge concentration. Then the practical spacing E_L and the burden V_L can be evaluated.

Fixation Factor

In the formulas, different fixation factors, f, are used for calculating the burden in different situations. For example: f = 1 in bench blasting with vertical holes positioned in a row with a fixed bottom. If the holes are inclined, it becomes easier to loosen the toe. To account for this, a lower fixation factor (f < 1) is used for an inclined hole. This results in a larger burden. In tunneling, a number of holes are sometimes blasted with the same delay number. Sometimes the holes have to loosen the burden upwards and sometimes downwards. Different fixation factors are used to include the effects of multiple holes and of gravity.

Stoping Holes

The method for calculating the stoping holes in sections B and C (Figure A-3) does not differ much from the calculation of the lifters. For stoping holes breaking horizontally and upwards in section B, a fixation factor, f, of 1.45 and an E/V ratio equal to 1.25 is used. The fixation factor for stoping holes breaking downwards is reduced to 1.2 and E/V-ratio should be 1.25.

The column charge concentration for both types of stoping holes should be equal to 50 percent of concentration of the bottom charge.

Contour Holes

If smooth blasting is not necessary, the burden and spacing of the contour holes is calculated according to what has been said about the lifters in section E, with the following exceptions:

a. fixation factor f = 1.2
b. E/V-ratio should be 1.25
c. charge concentration for the column charge is 50% of the bottom charge concentration .

The blast damaged roof and walls in a drift often need an excessive amount of support. In low strength rock, a long stand-up time usually can be achieved by more careful contour blasting. A 3-m long borehole with ANFO (1.5 kg/m) is capable of producing a damaged zone having a 1.3 - 1.2-m radius.

With smooth blasting, this damage zone is reduced to a minimum. Experience shows that the spacing is a linear function of hole diameter [5], or

$$E = k\ d \qquad \text{(m)} \qquad \text{(A-29)}$$

where the constant k is in the range of 15-16. An E/V ratio of 0.8 should be used.

For a 41 mm hole diameter, the spacing will be about 0.6 m and the burden about 0.8 m.

The minimum charge concentration per meter of borehole is also a function of the hole diameter. For hole diameters up to 0.15 m, the relationship

$$\ell = 90\ d^2 \qquad \text{(kg/m)} \qquad \text{(A-30)}$$

applies.

In smooth blasting, the total hole length must be charged to avoid ripping.

In Figure A-10, ℓ is plotted as a function of d.

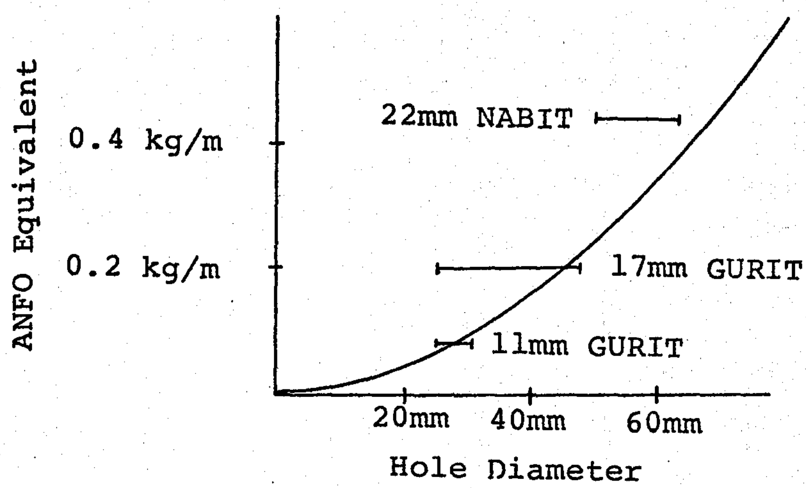


Figure A-10. Minimum Required Charge Concentration for Smooth Blasting and Recommended Practical Hole Diameter for NABIT and GURIT Charges

ROCK DAMAGE

The sudden expansion caused by an explosion in a borehole generates a stress wave that propagates into the rock mass. For an elastic material, the generated stress is directly proportional to density, particle velocity, and wave propagation velocity.

Close to the charge, the strain will reach a magnitude where permanent damage is produced. Whether this damage will have any significant influence on the stand-up condition for a tunnel depends upon the character of the damage, the exposure time, the influence of ground water, and last, but not least, the orientation of the joint planes with respect to the contour and the static load.

For a long time, the damage criteria for structures built in the vicinity of a blasting site have been based upon the peak particle velocity.

At Sve De Fo (Swedish Detonic Research Foundation), the same criterion has been found to apply for estimating the rock damage [4,6,9,10].

The empirical equation

$$v = 700\ Q^{0.7}/R^{1.5} \qquad \text{(A-31)}$$

where

v = the particle velocity (mm/s)

Q = the charge weight (kg)

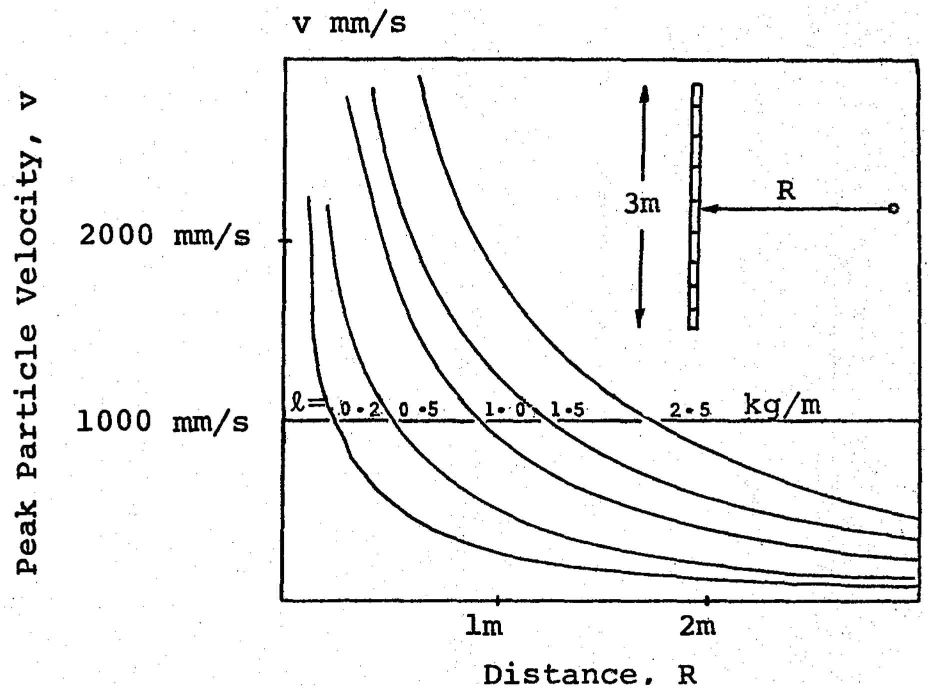
R = the distance (m)

was developed. It is valid for calculating the particle velocity at such distances where the charge can be treated as being spherical. For short distances, the discrepancy between the calculated and the measured values is unacceptable.

By performing an integration over the charge length, it was found possible to obtain the particle velocity as a function of distance, charge length, and charge concentration per meter of borehole. In Figure A-11, the results for a 3-m long charge are shown.

When the particle velocity exceeds some value between 700 and 1,000 mm/s (Figure A-11), cracks are induced or enlarged in a granite rock mass. A concentration of 1 kg/m means that damage occurs in a zone of radius 1.0 - 1.4 m around the charge.

In field experiments, very good agreement was found between the calculated and measured values for gneiss, pegmatite, and granite. Reports about damage zones also agree well with the calculated distances for similar charges if the 700-1,000 mm/s criterion is used. This is valid for charge conentration in the range of 0.2-75 kg/m.



<u>**Figure A-11. Peak Particle Velocity as a Function of Distance and Concentration for a 3m Long Charge**</u>

In the field experiments, accelerometers have been used together with FM-tape and transient recorders. Numerical integration provided the particle velocities. The closest distance from the charges located in 25 - 250 mm holes to the accelerometers has been in the range 1.5 - 13 m.

Measurements close to tunnel contours have indicated that charges in the row next to the contour often cause higher particle velocities and more damage than the smooth blasted (outer) row. If a smooth blasting result is not to be ruined by the rest of the holes, it is a good idea to reduce the charge concentration in the row next to the contour. Figure A-11 provides a guide for estimating the charge concentration. A concentration of 0.2 kg/m in the contour results in a damage zone of 0.3 m. If the burden was 0.8, one can see that the charge concentration for the inner row should not be limited to about 1 kg/m if the damage zone of 0.3 m is not to be exceeded.

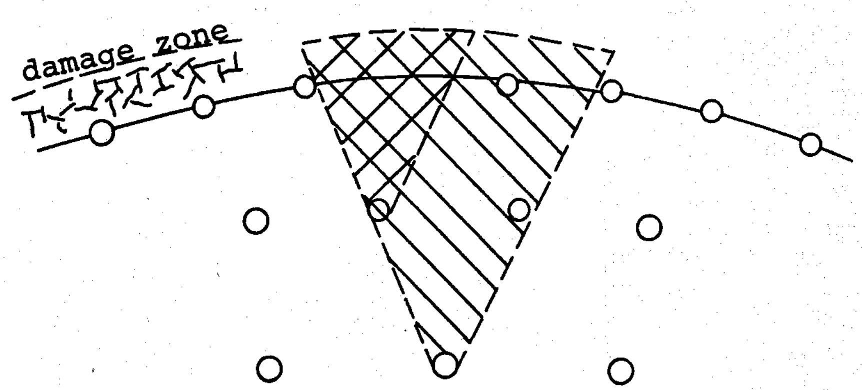


Figure A-12. A Well Designed Round Where the Charge Concentrations in the Holes Close to the Contour are Adjusted so that the Damage Zone from Each Hole Coincides

EXAMPLE OF CHARGE CALCULATIONS

Conditions

Hole diameter = 45 mm

Empty hole, Ø = 102 mm

Tunnel width = 4.5 mm

Abutment height = 4.0 m

Height of arch = 0.5 m

Smooth blasting in the roof

Lookout for contour holes ℓ = 3°

Angular deviation α = 1 cm/m

Collar deviation = 2 cm

Explosive: A watergel explosive is used with cartridge dimensions of Ø 25 x 600, Ø 32 x 600, Ø 38 x 600 mm

Heat of explosion = 4.5 MJ/kg

Gas volume at STP = 0.85 m^3/kg

Density = 1.200 kg/m^3

Rock constant c = -0.4

Calculation: Weight strength relative to LFB (equation A-1)

$$s_{LFB} = \frac{5x4.5}{6x5.0} + \frac{1x0.85}{6x0.85} = 0.92$$

and

$$s_{ANFO} = 0.92/0.84 = 1.09$$

Charge concentration

Ø(mm)	ℓ(kg/m)
25	0.59
32	0.97
38	1.36

Advance: Using Ø = 102 mm equation (A-2) results in a hole depth of 3.2 m and an advance of 3.0 m.

<u>Cut</u>:

First Quadrangle:

Maximum burden: $V = 1.7\emptyset = 0.17$ m

Practical burden: $V_1 = 0.12$ m (equation A-6)

Charge concentration: $\ell = 0.58$ kg/m (equation A-9)

ℓ or the smallest cartridge is 0.59 kg/m which is sufficient for clean blasting the opening.

Unloaded hole length = 10d = 0.45 m

(equation A-19)

Hole distance in quadrangle $B' = \sqrt{2}\ V_1 = 0.17$ m

No. of Ø 25x600 cartridges = (3.2-0.45)/0.6 = 4.5

Second Quadrangle:

The rectangular opening towards which to blast is

$B = \sqrt{2}\ (0.12-0.05) = 0.10$ m (equation A-12)

Maximum burden for Ø25 cartridges

$V = 0.17$ m (equation A-11)

Maximum burden for Ø cartridges

$V = 0.21$ m (equation A-11)

Equation (A-15) says the practical burden must not exceed 2B. This implies that the Ø32x600 cartridges are the most suitable ones in this quadrangle.

Practical burden: $V_2 = 0.16$ m (equation A-14)

Unloaded hole length: $h = 0.45$ m (equation A-19)

Hole distance in quadrangle: $B' = \sqrt{2}\ (0.16+0.17/2) = 0.35$ m

Number of Ø32x600 cartridges = 4.5

Third Quadrangle:

$B = \sqrt{2}\ (0.16+0.17/2-0.05) = 0.28$ m

Use Ø38x600 cartridges with charge concentration $\ell = 1.36$ kg/m

Maximum burden: $V = 0.42$ m

Practical burden: $V_3 = 0.37$ m

Unloaded hole length: $h = 0.45$ m

Hole distance in quadrangle: $B' = \sqrt{2}\ (0.37+0.35/2) = 0.77$ m

Number of Ø38x600 cartridges = 4.5

Fourth Quadrangle:

$B = \sqrt{2}\ (0.37+0.35/2-0.05) = 0.70$ m

Maximum burden: $V = 0.67$ m

Practical burden: $V_4 = 0.62$ m

Unloaded hole length: $h = 0.45$ m

$B' = \sqrt{2}\ (0.62+0.77/2) = 1.42$ m

Number of Ø38x600 cartridges = 4.5

The side length of this quadrangle is 1.42 m which is comparable to the square root of the advance.

Therefore, there is no need for more quadrangles.

Lifters

Use Ø38x600 cartridges with a charge concentration of $\ell = 1.36$ kg/m

Maximum burden: $V = 1.36$ m (equation A-20)

Number of lifters: $N = 5$ (equation A-23)

Spacing: $E_L = 1.21$ m (equation A-24)

Spacing, corner holes: $E^1L = 1.04$ m (equation A-25)

Practical burden: $V_L = 1.14$ m (equation A-26)

Length of bottom charge: $h_b = 1.43$ (equation A-27)

Length of column charge: $h_c = 1.32$ m (equation A-28)

This charge concentration shall be 70 percent of the bottom charge concentration: 0.70x1.36=0.95 kg/m. Use 2.5 cartridges Ø38x600 as the bottom charge and two cartridges Ø32x600 as the column charge.

Contour Holes, Roof

Smooth blasting with Ø25x60x600 cartridges is specified.

Spacing: $E = 0.68$ m (equation A-29)

Burden: $V = E/0.8 = 0.84$ m

Due to lookout and faulty drilling, the practical burden becomes: $V_R = 0.84-3.2 \sin 3°-0.05 = 0.62$ m. The minimum charge concentration for this smooth blasting is $\ell = 90\ d^2 = 0.18$ kg/m (equation A-29)

The charge concentration for the Ø25x600 cartridges is 0.59 kg/m, which is considerably more than that which is actually needed.

Number of holes: integer of (4.7/0.68+2) = 8

Five cartridges per hole are used.

Contour Holes, Wall

The abutment height is 4.0 m and from the calculation above, it is known that the lifters should have a burden of 1.14 m, and the roof holes should have a burden of 0.62 m. This implies that there is 4.0-1.14-0.62 = 2.24 m left in the contour along which to position the wall holes. By using a fixation factor f = 1.2, and an E/V ratio equal to 1.25, equation (A-20) results in a maximum burden: V = 1.33 m.

Practical burden: $V_W = 1.33 - 3.2 \sin 3° - 0.05 = 1.12$ m.

Number of holes = integer of (2.24/(1.33•1.25 + 2) = 3

Spacing = 2.24/2 = 1.12 m

Length of bottom charge: $h_b = 1.40$ m

Length of column charge: $h_c = 1.35$ m

Two-and-one-half cartridges Ø38x600 are used as the bottom charge, and two cartridges Ø32x600 are used in the column.

Stoping

The side of the fourth quadrangle in the cut is 1.42 m and the practical burden, V_W, for the wall holes was determined to be 1.12 m. As the tunnel width is 4.5 m, a distance of 4.5-1.42 - 2•1.12 = 0.48 m is available for placing horizontal stoping holes.

Maximum burden: (f = 1.45) V = 1.21m

Practical Burden: $V_H = 1.21 - 0.05 = 1.16$m

Instead the burden $V_H = 0.84$m due to the tunnel geometry. The height of the fourth quadrangle was 1.42m and this will of course determine the spacing for the two holes, which becomes = 1.42m.

For stoping downwards:

Maximum burden: V = 1.33m

Practical burden: $V_D = 1.28$m

The maximum height of the tunnel is specified to be 4.5 m. If the height of the fourth quadrangle (1.42 m), the burdens for the lifters (1.14 m), and the roof holes (0.62 m) are subtracted, there is 1.32 m left for a stoping hole. This is just a little more than the practical burden, but if the stoping holes are placed at 1.28 m above the cut, the remaining 0.04 m will in all probability be removed by the overcharged contour. Furthermore, the formulas used in the calculation have a safety margin that can tolerate small deviations. Three holes for stoping downwards are positioned above the fourth quadrangle. The charge distribution for the stoping holes is the same as for the wall holes.

This work was done as part of the rock blasting research program of the Swedish Detonic Research Foundation, Sve De Fo, supported by Swedish Industry and the Swedish Board for Technical Development. The author gratefully acknowledges professional discussions with present and former colleagues at Sve De Fo and Nitro Consult whose experiences in the art of charge calculations helped in forming this appendix.

APPENDIX B
EXPLOSIVE CONSUMPTION AND LOADING PATTERNS USED IN THE EXPERIMENTAL ROOM

In this appendix, the loading of the different rounds is described. Every hole in the round has been assigned a unique number which makes it easy to determine exactly how the different types of holes have been loaded. Remarks have been written to describe where changes in the original design had to be made due to difficulties in drilling or loading. Figures B-1 through B-4 show the numbering convention in each round. Tables B-1 through B-6 indicate the number of sticks of explosive in each hole.

Round One

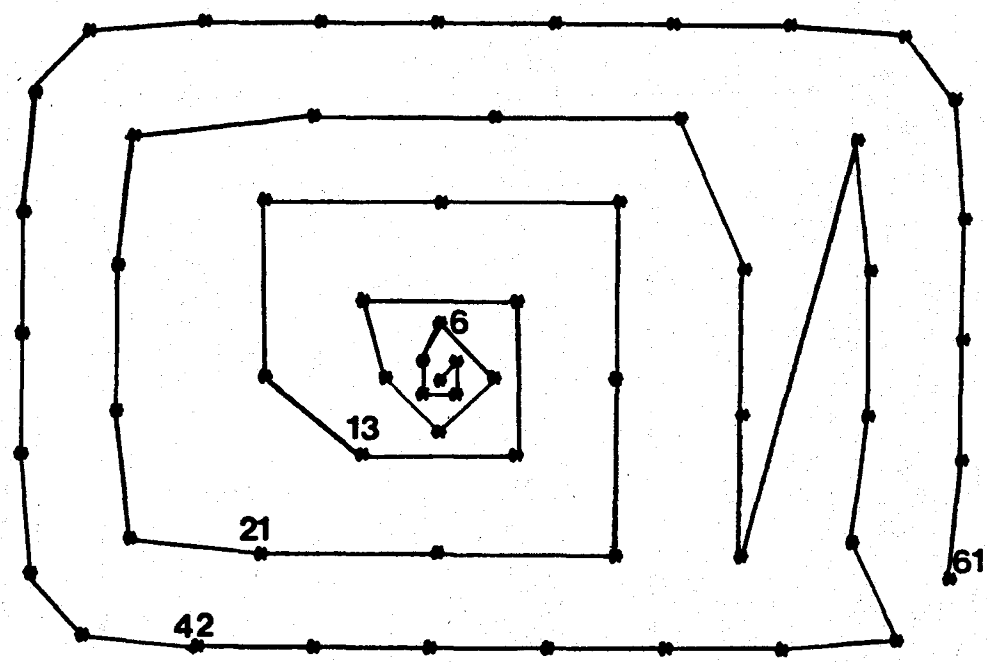


Figure B-1. Number Convention for All Holes in Round One

Table B-1. Explosive Loading for Round One

Hole No.	Tovex 100 1"x16"	Tovex 210 1-1/8"x16"	Tovex 220 1-1/4"x16"	PETN-cord 200 seismic
2- 5		5		
6-13			5	
14-21		3	2	
22-42	4	1		
43-61				3 x 2 m

Remarks: Hole four was hard to load in the correct manner, as it ran into the empty hole.

Holes 35 and 38 could only be loaded with three sticks 1" x 16" and one stick 1-1/8" x 16".

Hole 57 could not be drilled. Due to the large hole deviations, holes 14-21 and 36-42 had to be loaded heavier than the calculated amounts to break the burdens in a proper way.

Round Two

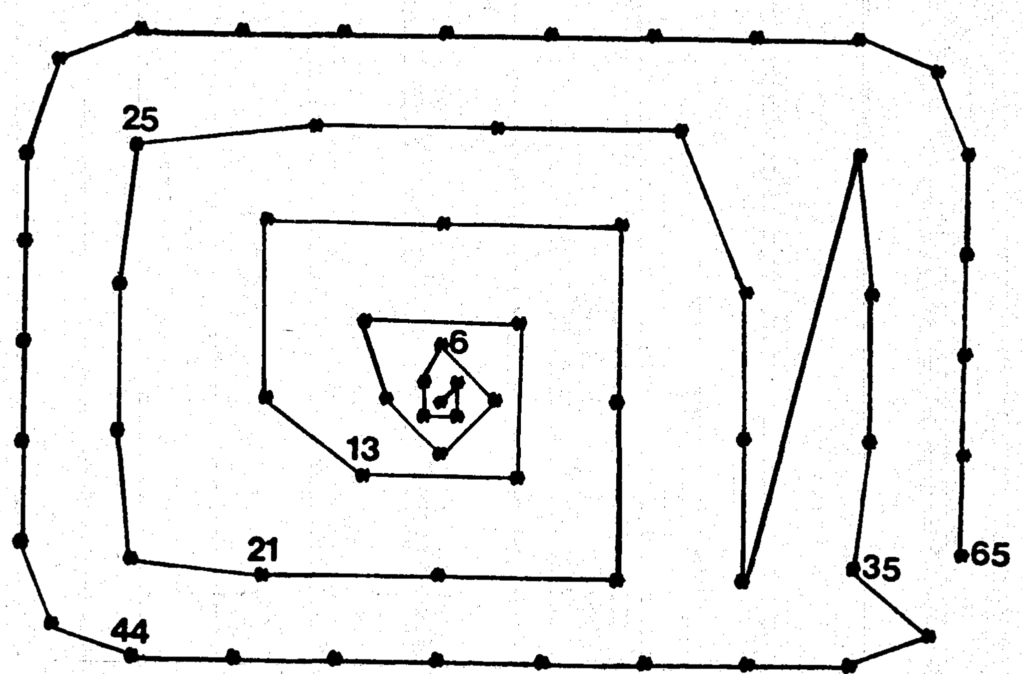


Figure B-2. Number Convention for the Holes in Round Two

Table B-2. Explosive Loading for Round Two

Hole No.	Tovex 100 1"x16"	Tovex 210 1-1/8"x16"	Tovex 220 1-1/4"x16"	PETN-cord 200 seismic
2- 5		5		
6-13			5	
14-21		5		
22-25	4	1		
26-35	3		2	
37-44	1			4 x 2 m
45-53	1/2			3 x 2 m
54-65,36	1			3 x 2 m

Remarks: The right rib was more heavily loaded due to hole deviations. One stick of Tovex was used in the bottom of the right rib holes instead of a half stick.

Round Three

The number convention for the holes is identical to the one for round two.

Table B-3. Explosive Loading for Round Three

Hole No.	Tovex 100 1"x16"	Tovex 210 1-1/8"x16"	Tovex 220 1-1/4"x16"	PETN-cord 200 seismic
2- 5		4.5		
6-13			5	
14-21		5		
22-35	4	1		
36-45	1			4 x 2 m
46-65	1/2			3 x 2 m

Remarks: When loading the round, problems occurred with the lifters. It was not possible to load holes 39 and 40 completely to the bottom due to mud, thus 0.5 m was left uncharged. Instead, two sticks of Tovex 100 were loaded in hole 38.

Round Four

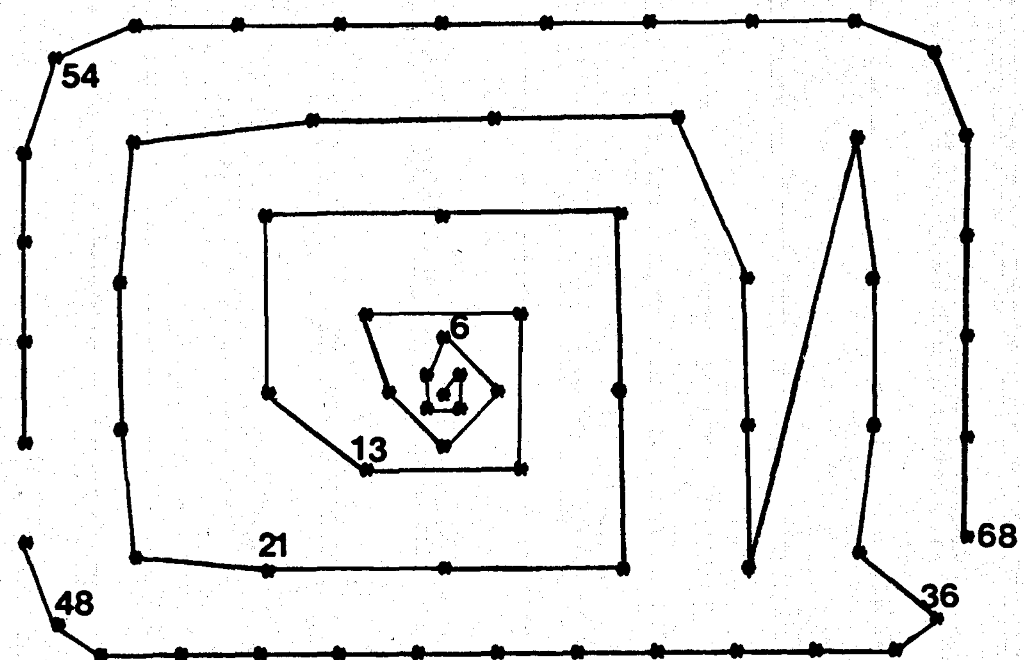


Figure B-3. Number Convention for the Holes in Round Four

Table B-4. Explosive Loading in Round Four

Hole No.	Tovex 100 1"x16"	Tovex 210 1-1/8"x16"	Tovex 220 1-1/4"x16"	PETN-cord 200 seismic
2- 5		5		
6-13			5	
14-21		5		
22-35	3	2		
36-68	1/2			3 x 2 m

Remarks: There were numerous problems with the drilling. Three rods got stuck in a joint positioned 1.2 m into the face at the left side. Therefore, hole 47 could not be drilled. Holes 36-48 were connected to each other in order to try to initiate the lifters simultaneously.

Round Five

The number convention is the same as for round four. The holes were also loaded exactly as in round four.

Remarks: All perimeter holes were connected to each other with a 50 grain PETN-cord in order to get simultaneously initiation. The lifters were initiated in one interval, the rib holes in a later interval, and the back holes in a final interval.

Round Six

The number convention is the same as round four.

Table B-4. Explosive Loading in Round Six

Hole No.	Tovex 100 1"x16"	Tovex 210 1-1/8"x16"	Tovex 220 1-1/4"x16"	Tovex T-1	PETN-cord 200 seismic
2- 5		6.5			
6-13			6.5		
14-21		6.5			
22-35	4.5	2			
37-41					
59-63	1/2				3 x 2.8 m
64-68,36					
42-58				2.9	

Remarks: In this round, Tovex T-1 was used for contour blasting in the left half of the round. The round has to be partly redrilled and reshot as the hole deviations were bad and the round misfired. The drilling crew which fired the round also tied the round in as one single series instead of a minimum of two parallel series. A lot of caps, therefore, didn't detonate.

Round Seven

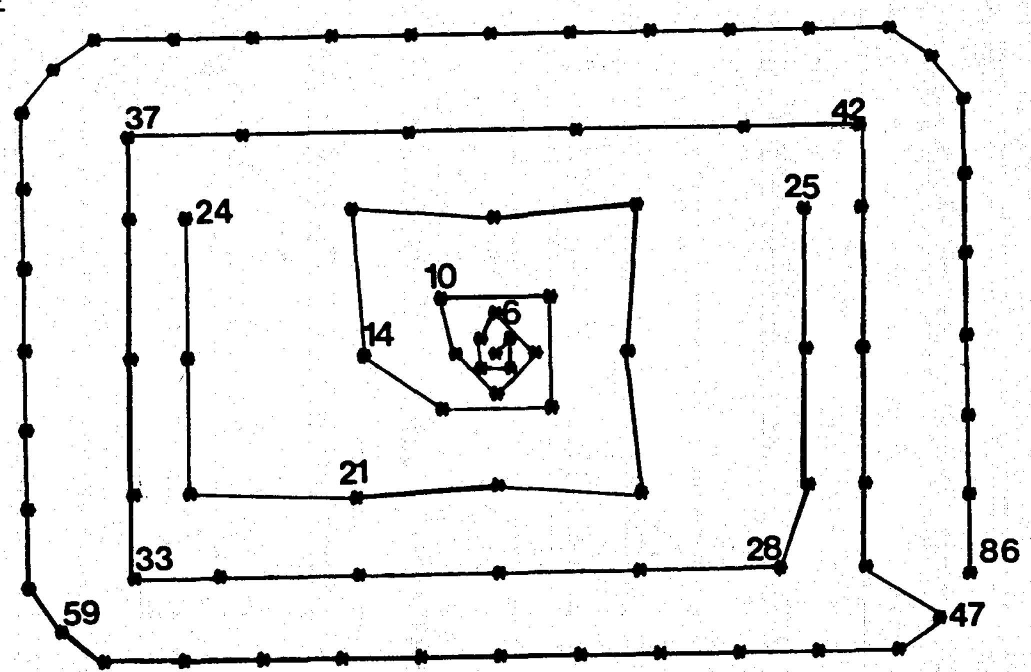


Figure B-4. Number Convention for the Holes in Round Seven

Table B-6. Explosive Loading in Round Seven

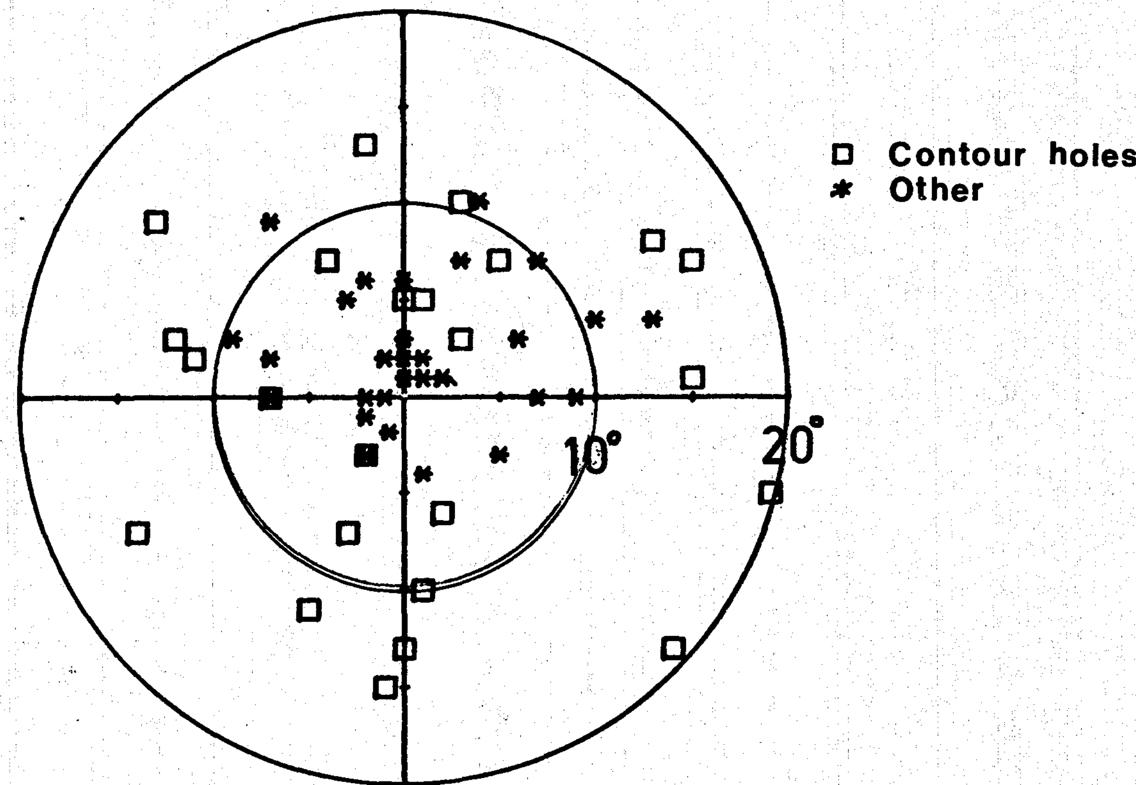
Hole No.	Tovex 100 1"x16"	Tovex 210 1-1/8"x16"	Tovex 220 1-1/4"x16"	PETN-cord 200 seismic
2- 5		5		
6-21			5	
22-27		3	2	
28-46	5			
48-58	1/2			3 x 2 m
59-86,47	1/2			2 x 2 m

Remarks: The round was designed for a hole depth of 3 m, but equipment failure forced us to drill only a 2.4-m round. The stoping rows (holes 28-46) next to the contour row had the same lookout angle as the contour row. For some reason, holes 68, 82, and 86 had not been drilled.

APPENDIX C
HOLE LOCATIONS AND HOLE DEVIATION

The information in this appendix is explained in Section 5 of the report. There are two drawings for each round on the first seven pages. The first is a computer plot of achieved lookout angles to be compared with Figure 5-1. The second drawing shows the achieved contour (dashed lines) versus designed contour.

Histograms and normal distribution curves are plotted for rounds one through six in the remaining pages of this appendix.



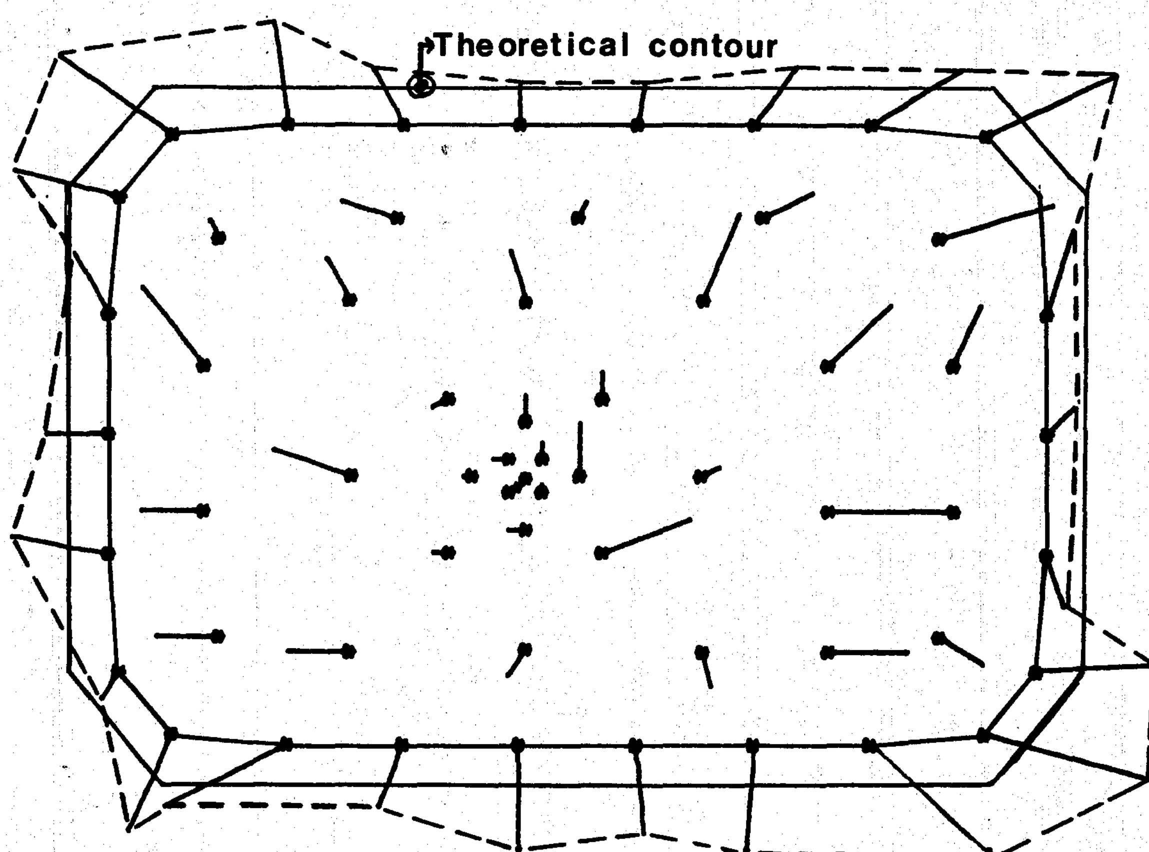


Figure C-1. Round 1 Hole Deviations and Achieved Contour

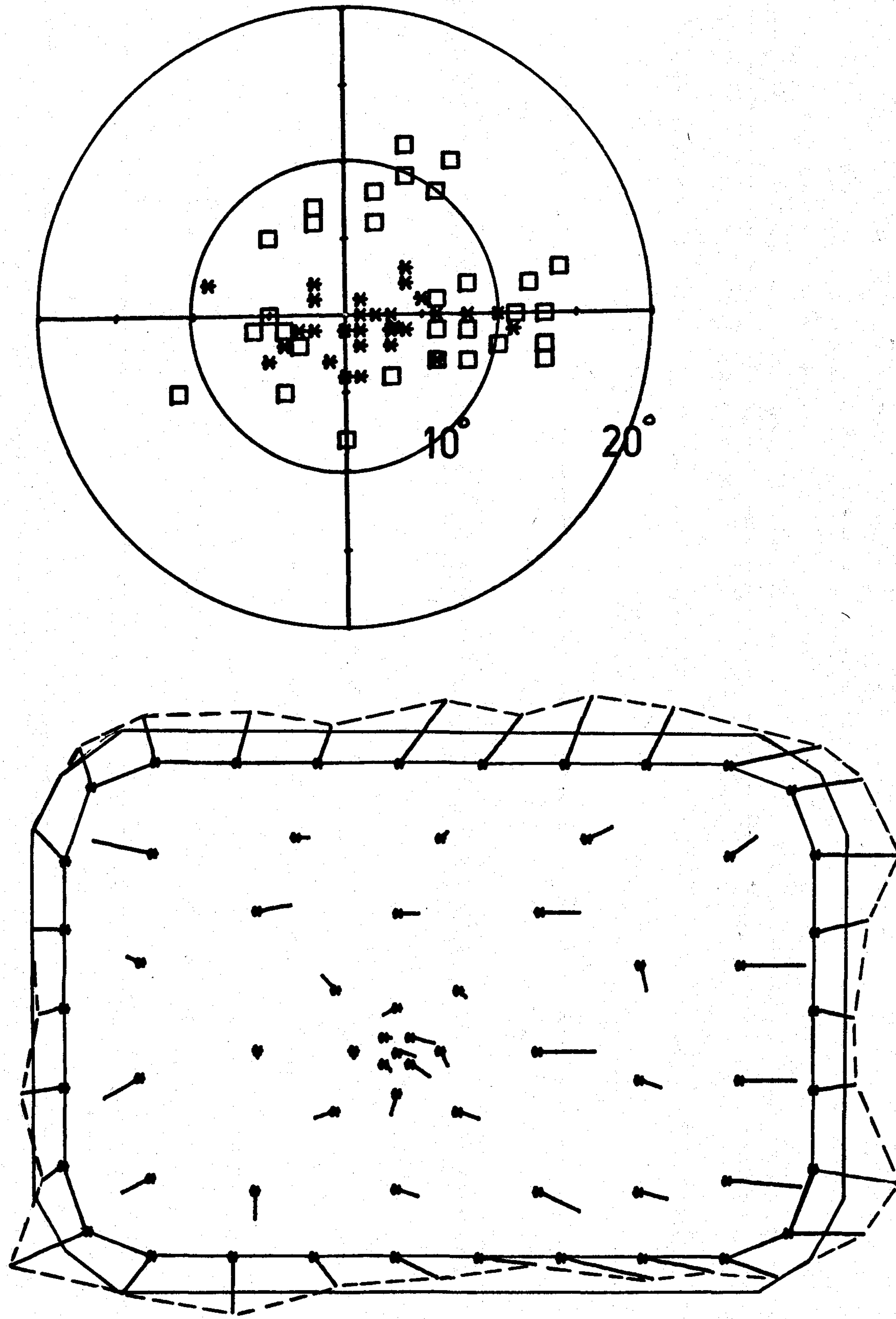
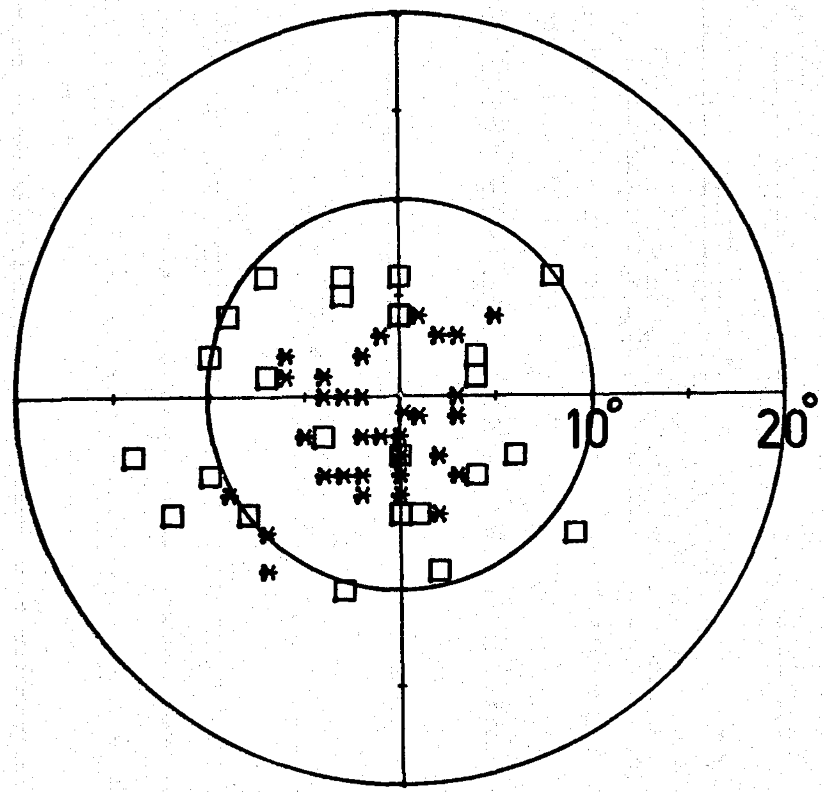


Figure C-2. Round 2 Hole Deviations and Achieved Contour



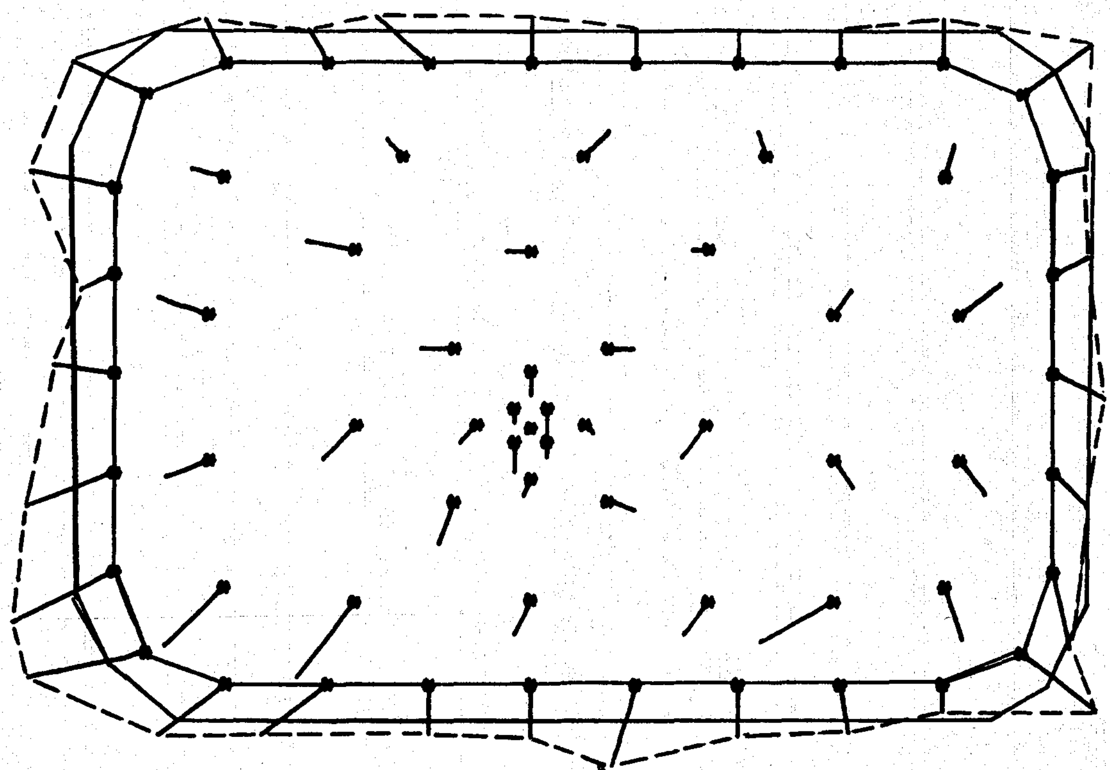


Figure C-3. Round 3 Hole Deviations and Achieved Contour

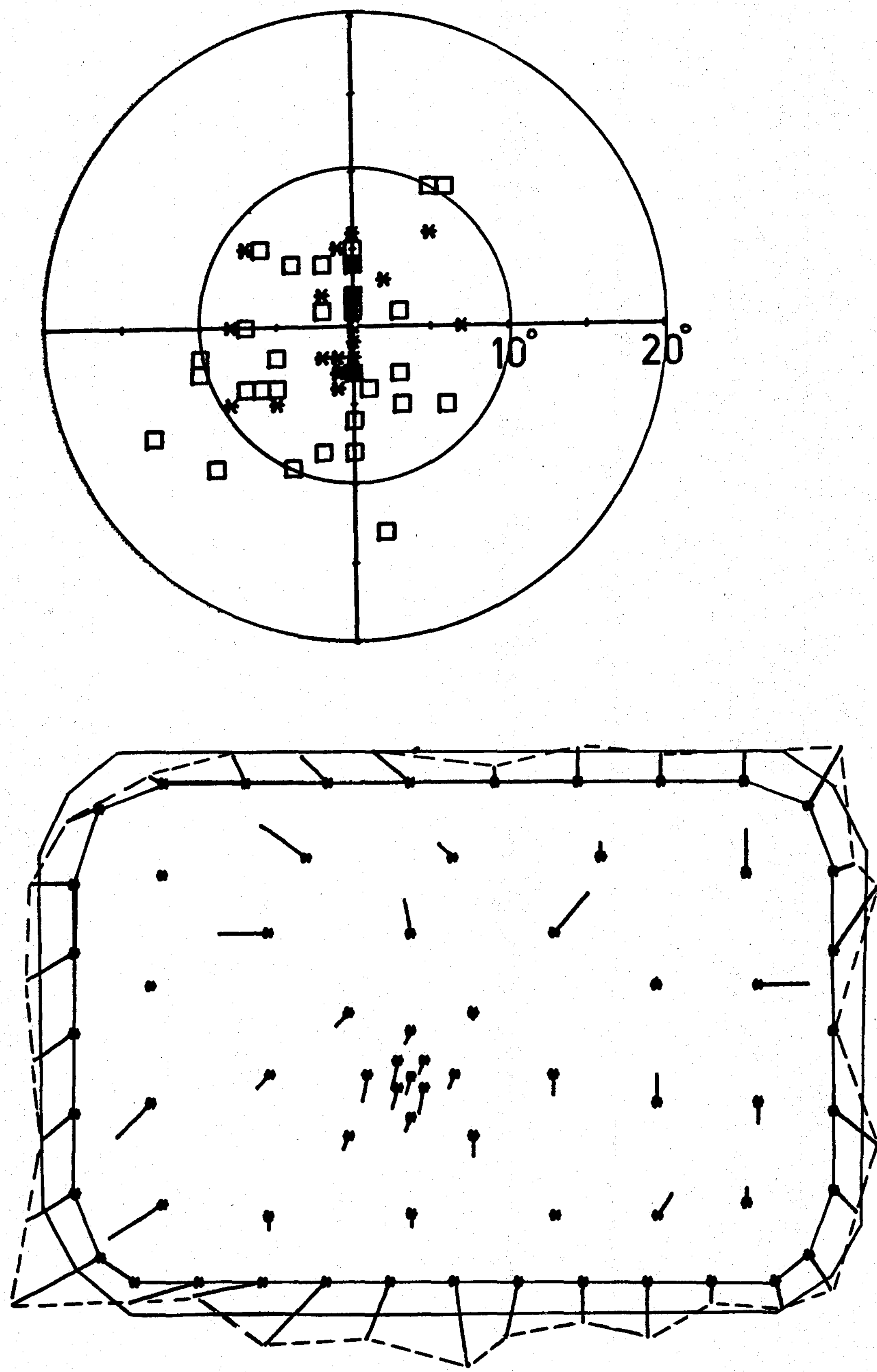


Figure C-4. Round 4 Hole Deviations and Achieved Contour

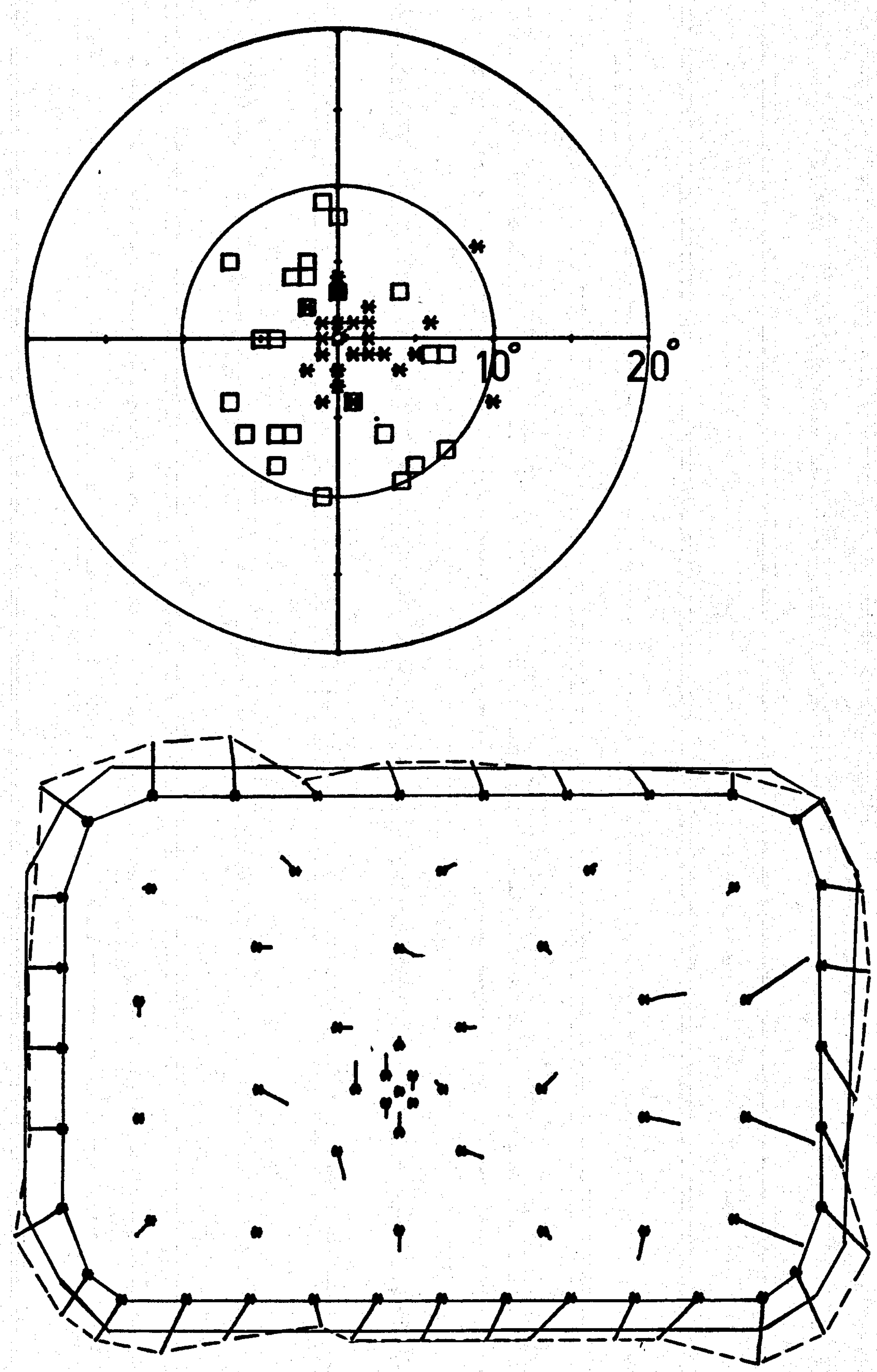
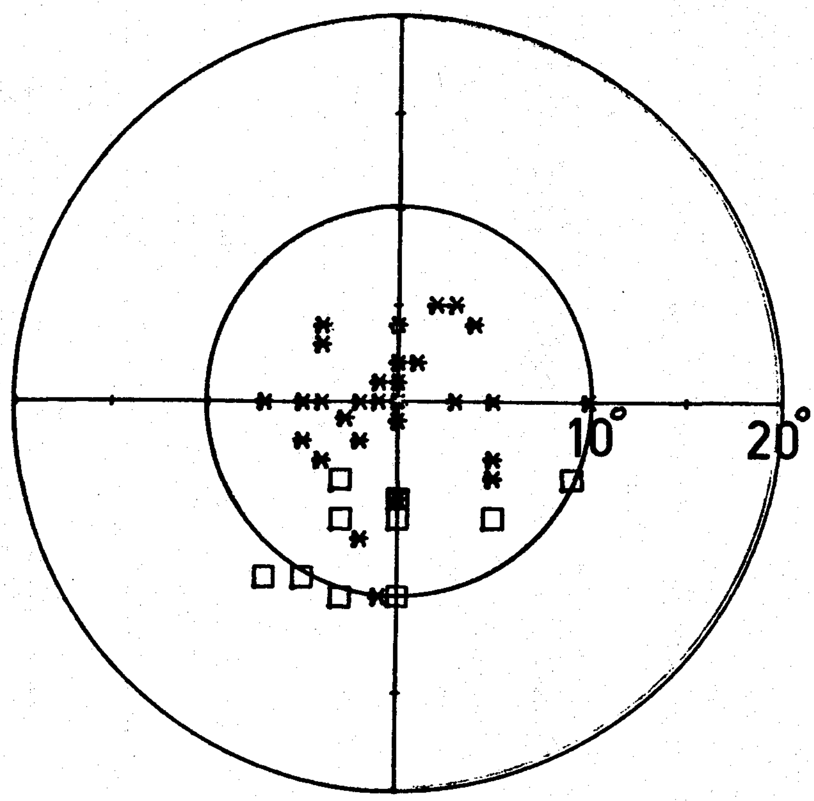


Figure C-5. Round 5 Hole Deviations and Achieved Contour



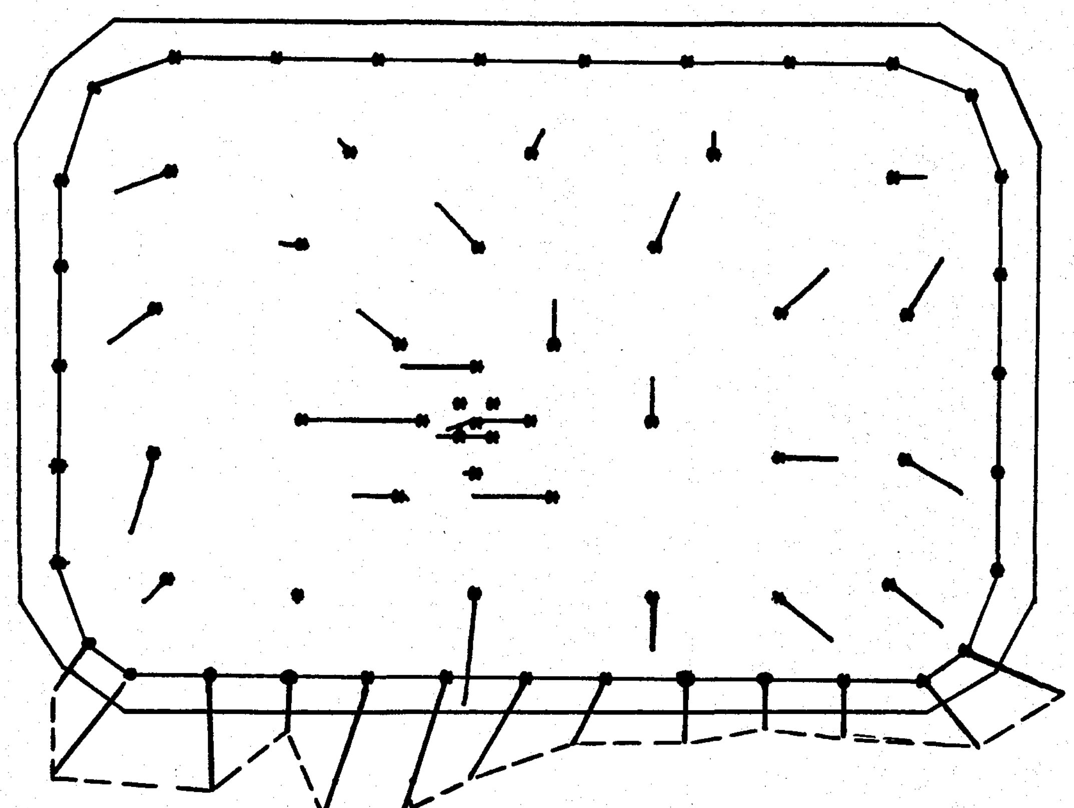


Figure C-6. Round 6 Hole Deviations and Achieved Contour

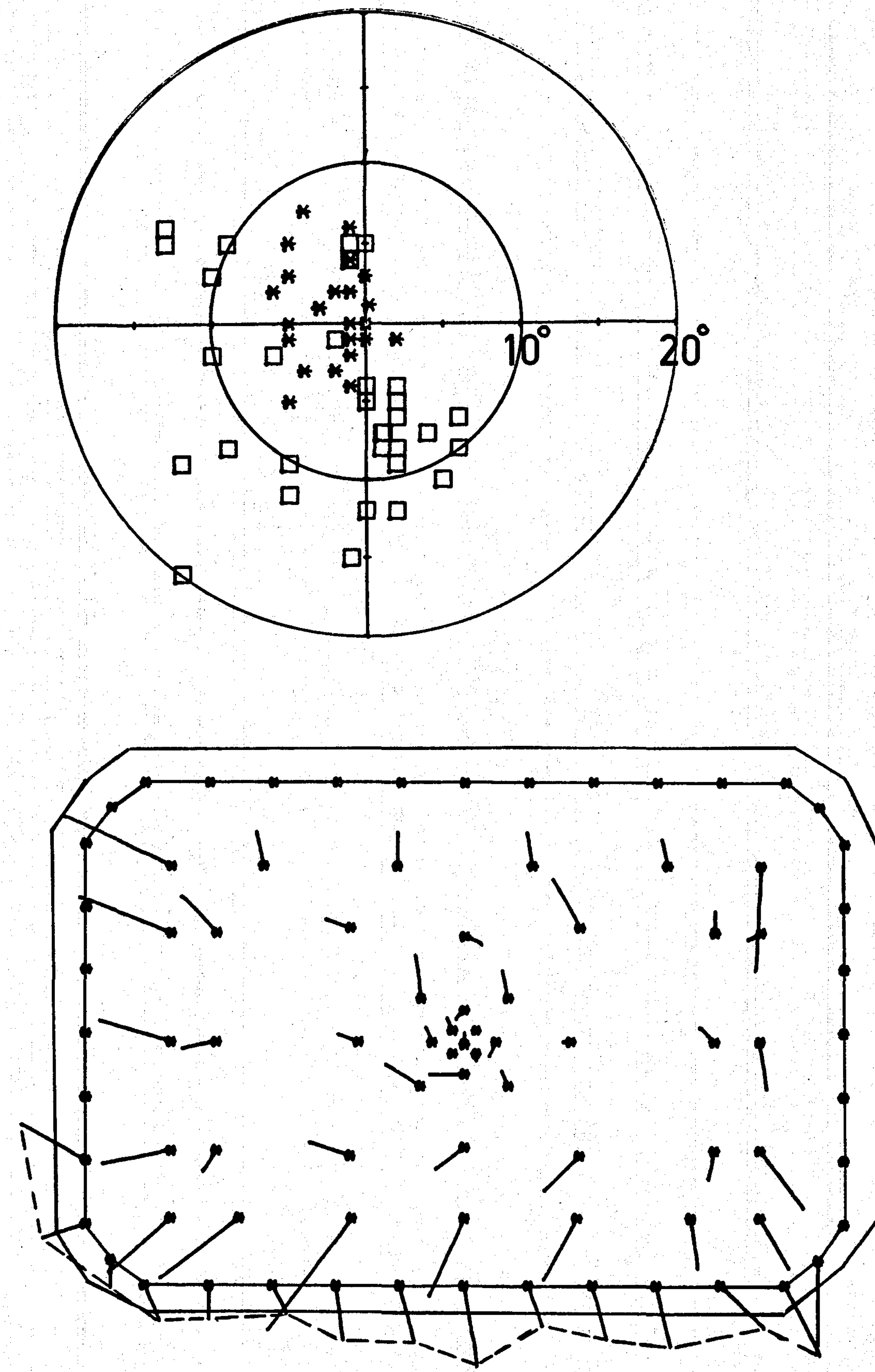
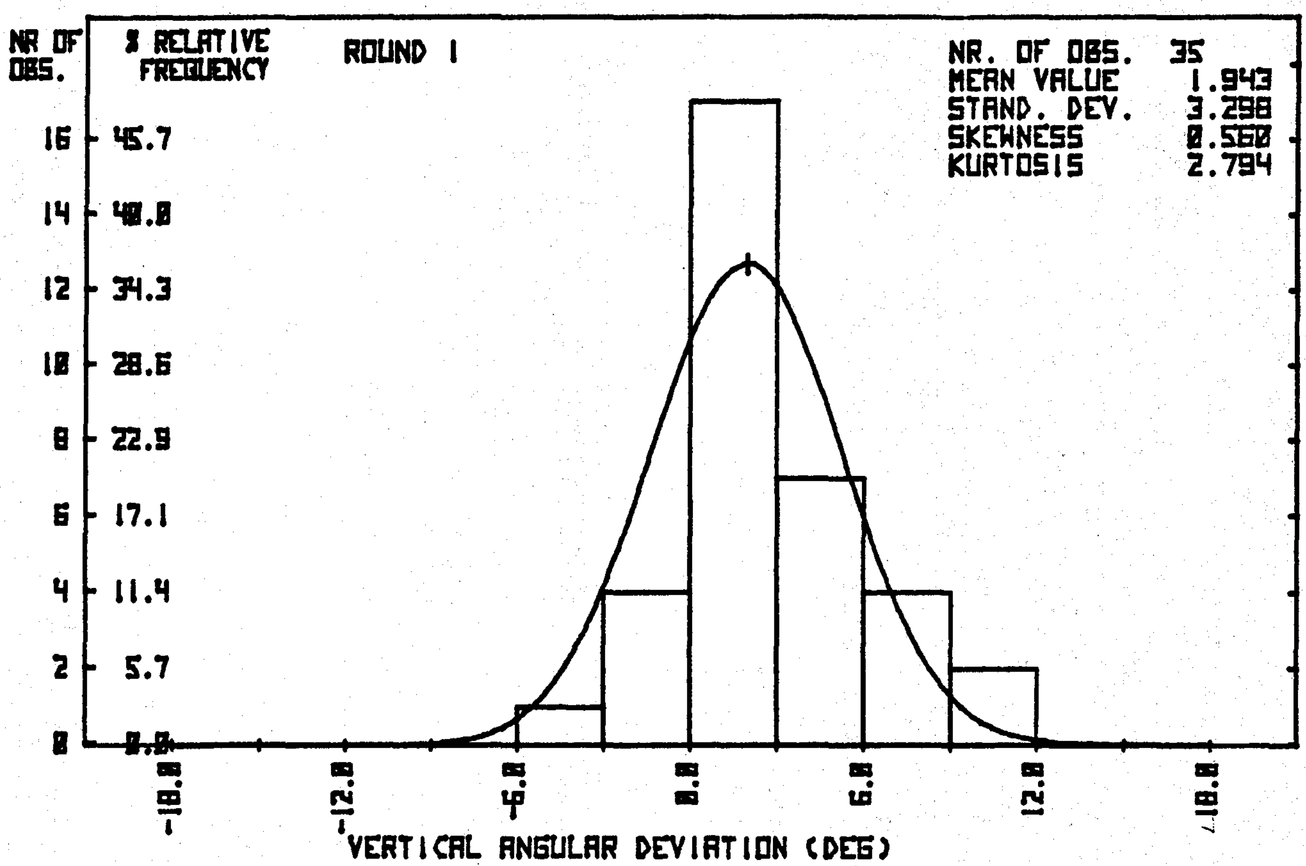


Figure C-7. Round 7 Hole Deviations and Achieved Contour



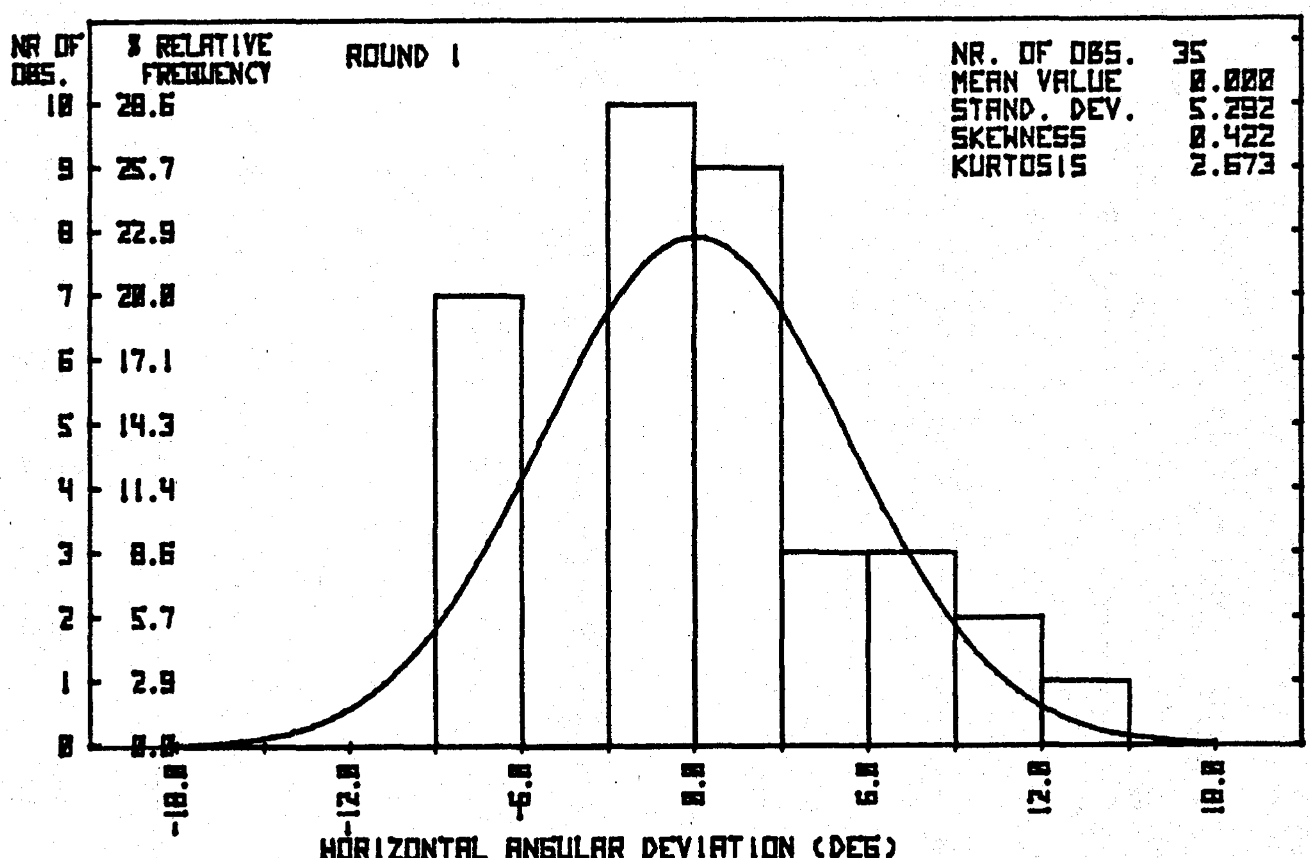
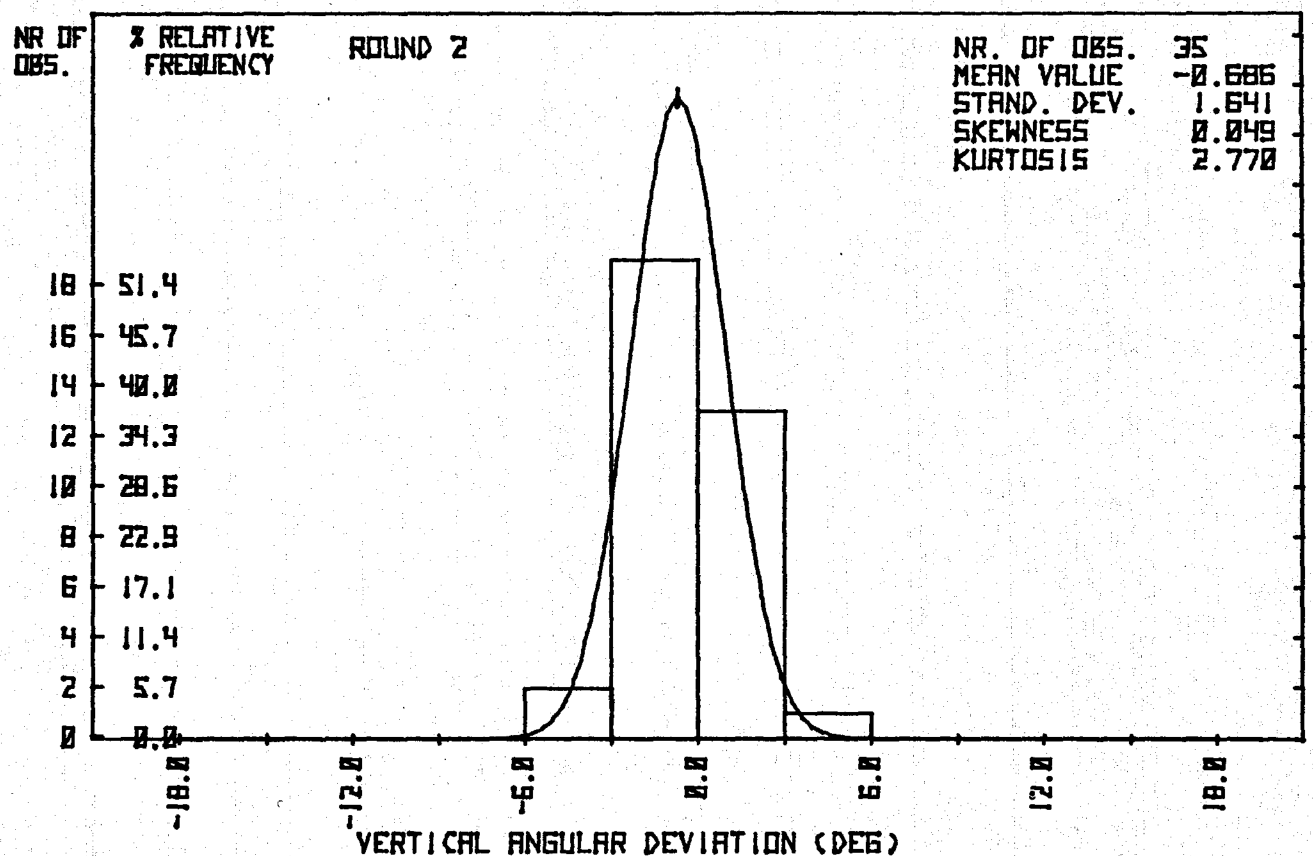


Figure C-8. Round 1 Vertical and Horizontal Deviations for Stoping and Cut Holes



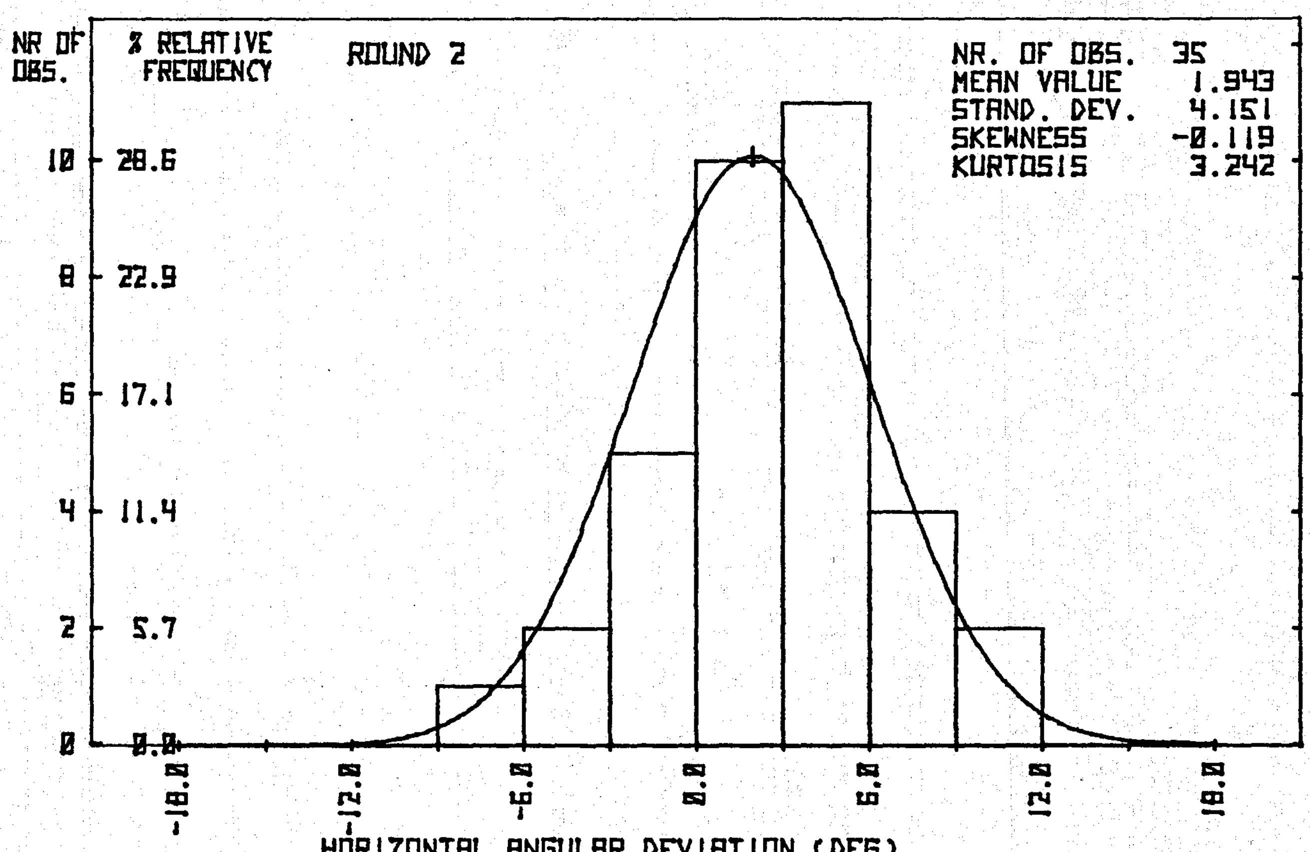
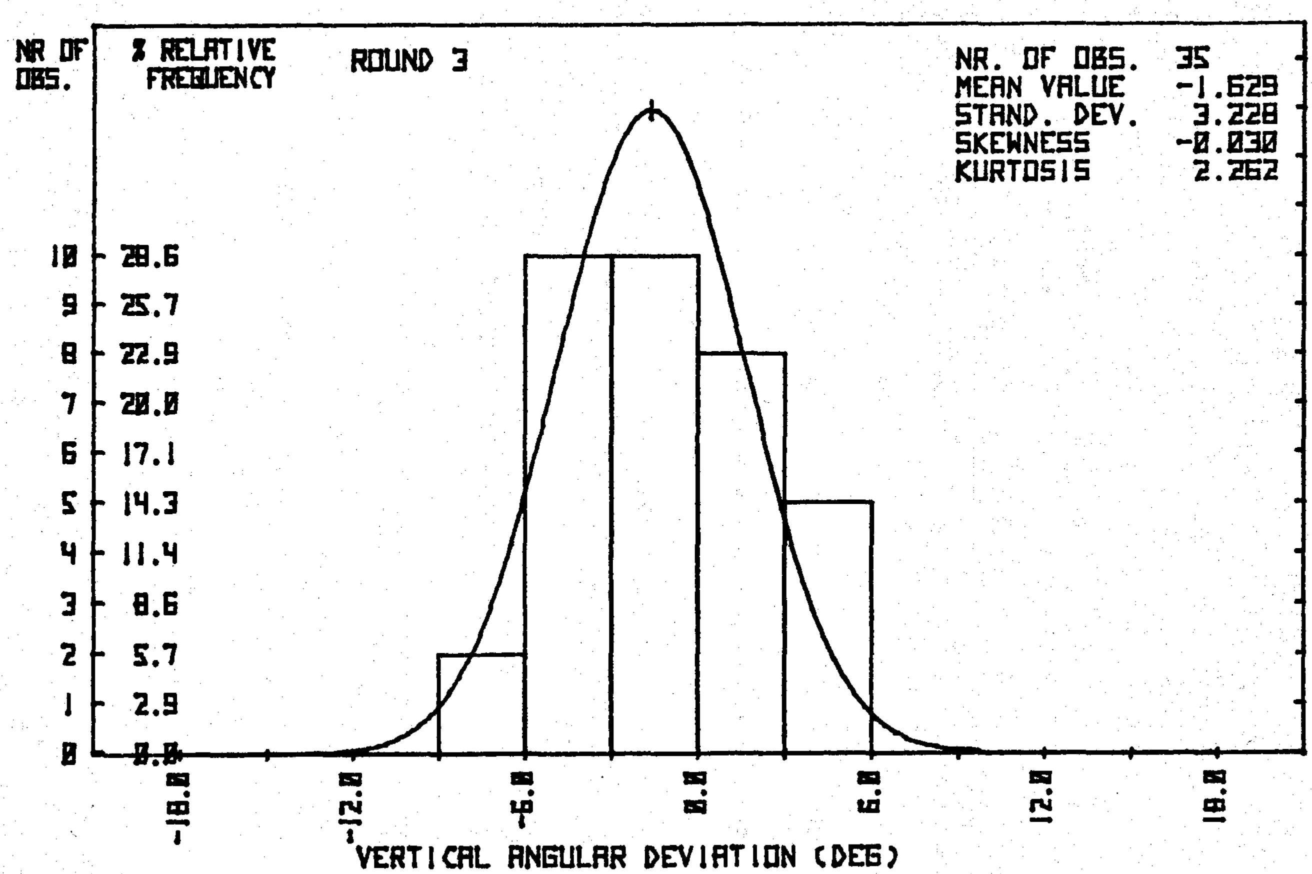


Figure C-9. Round 2 Vertical and Horizontal Deviations for Stoping and Cut Holes



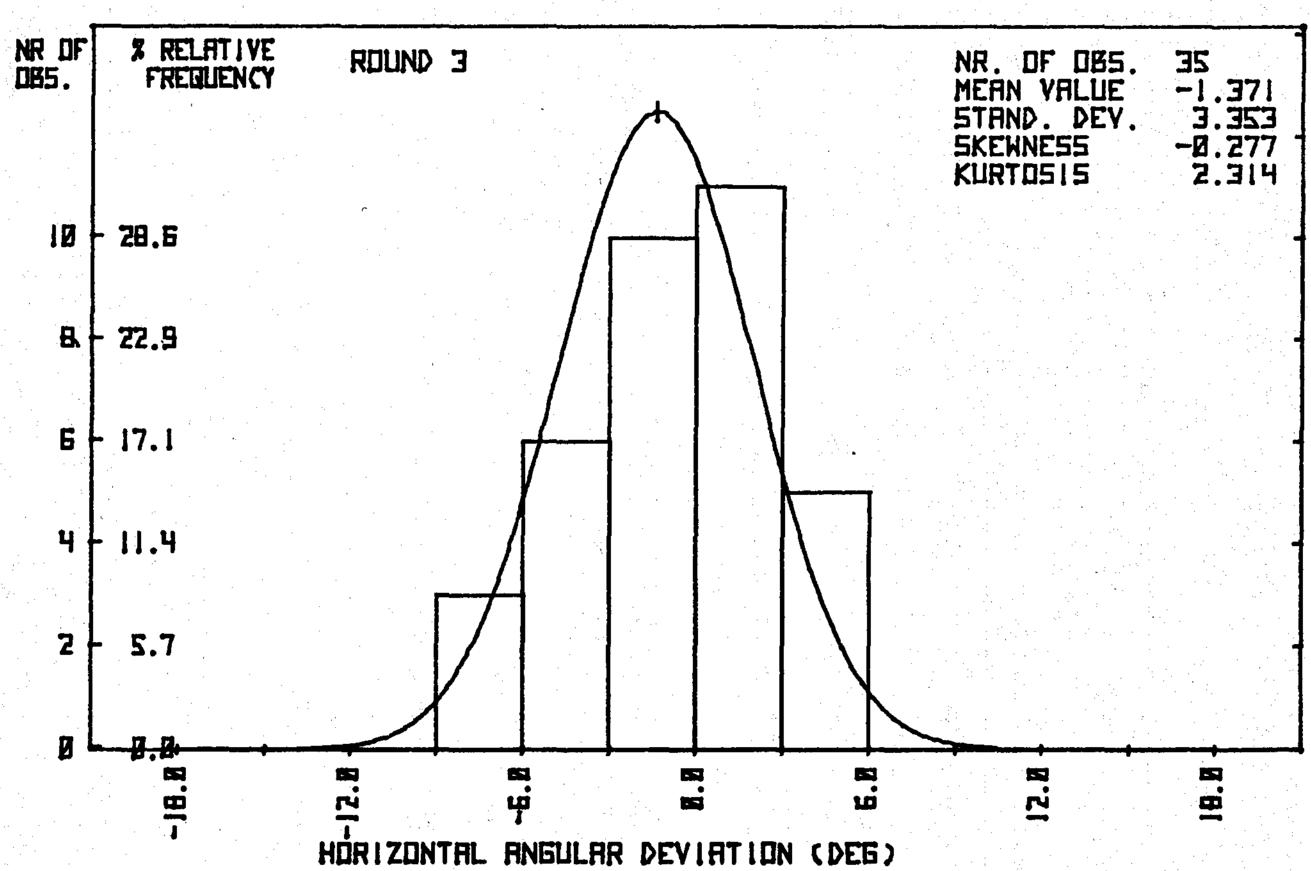
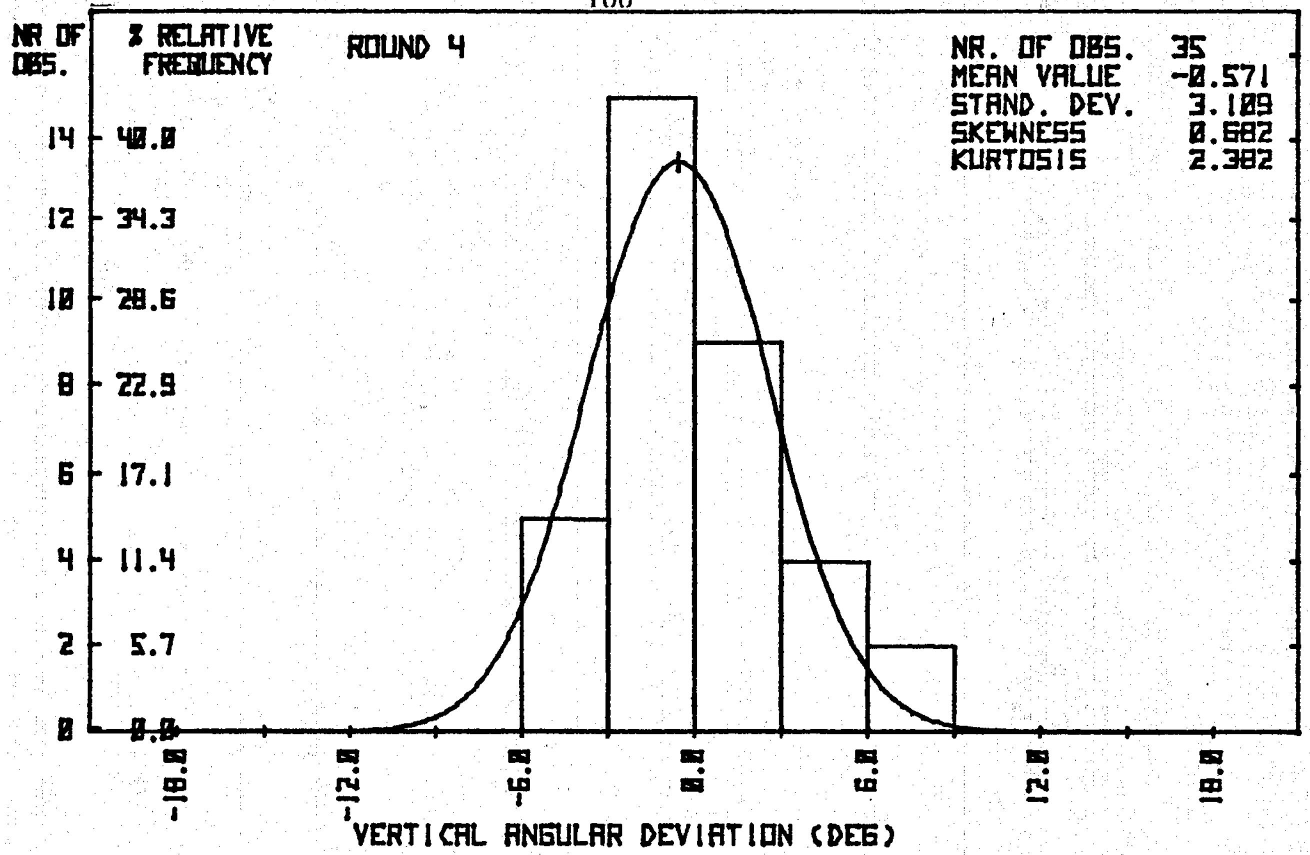


Figure C-10. Round 3 Vertical and Horizontal Deviations for Stoping and Cut Holes



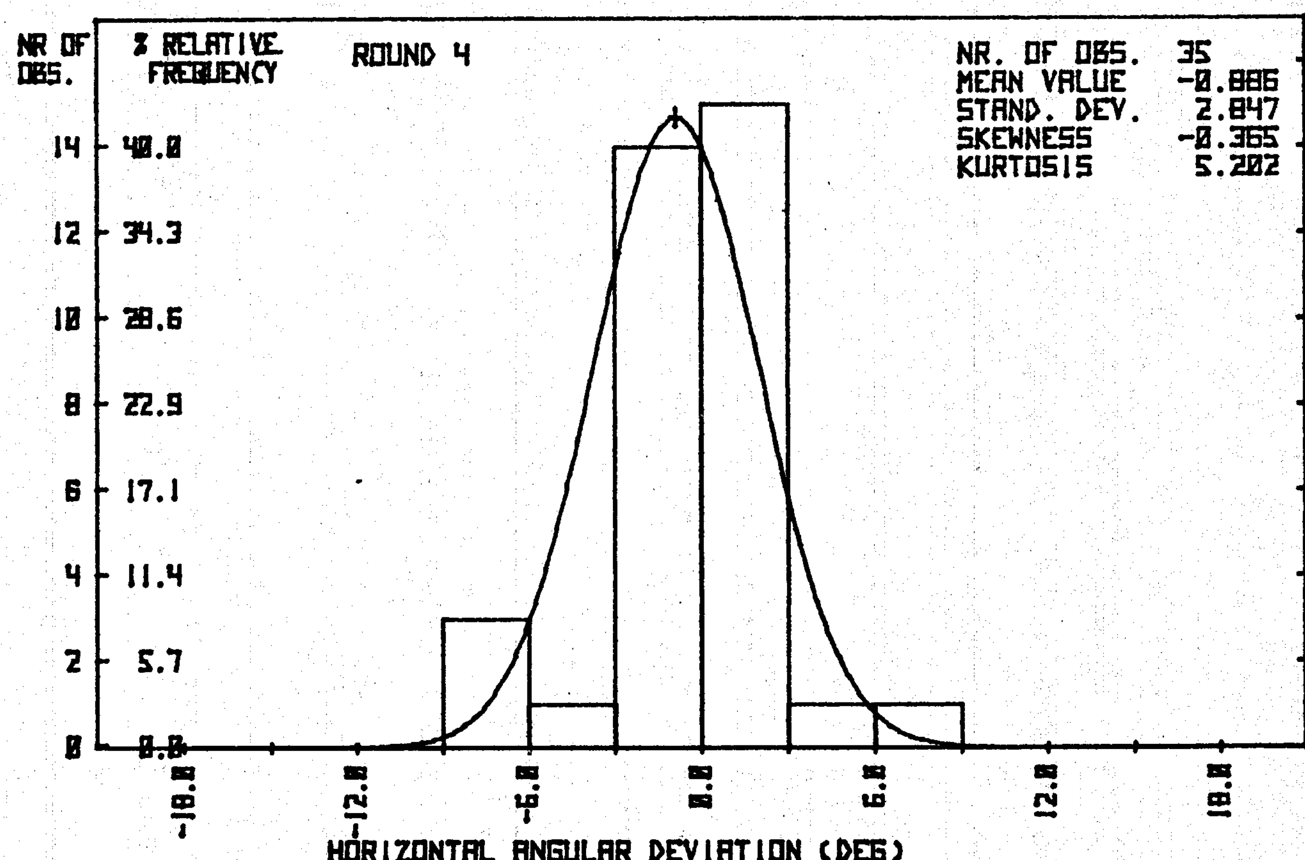
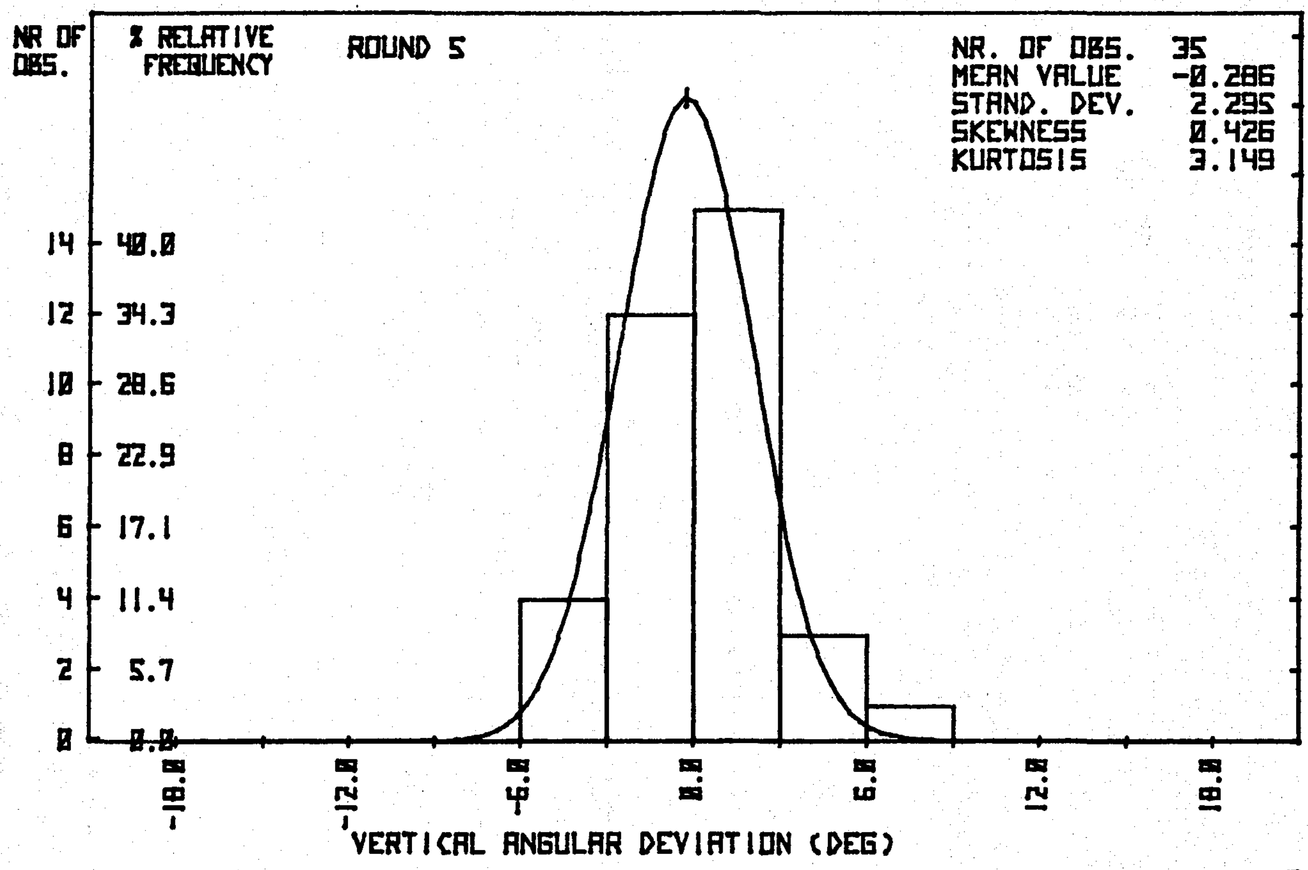


Figure C-11. Round 4 Vertical and Horizontal Deviations for Stoping and Cut Holes



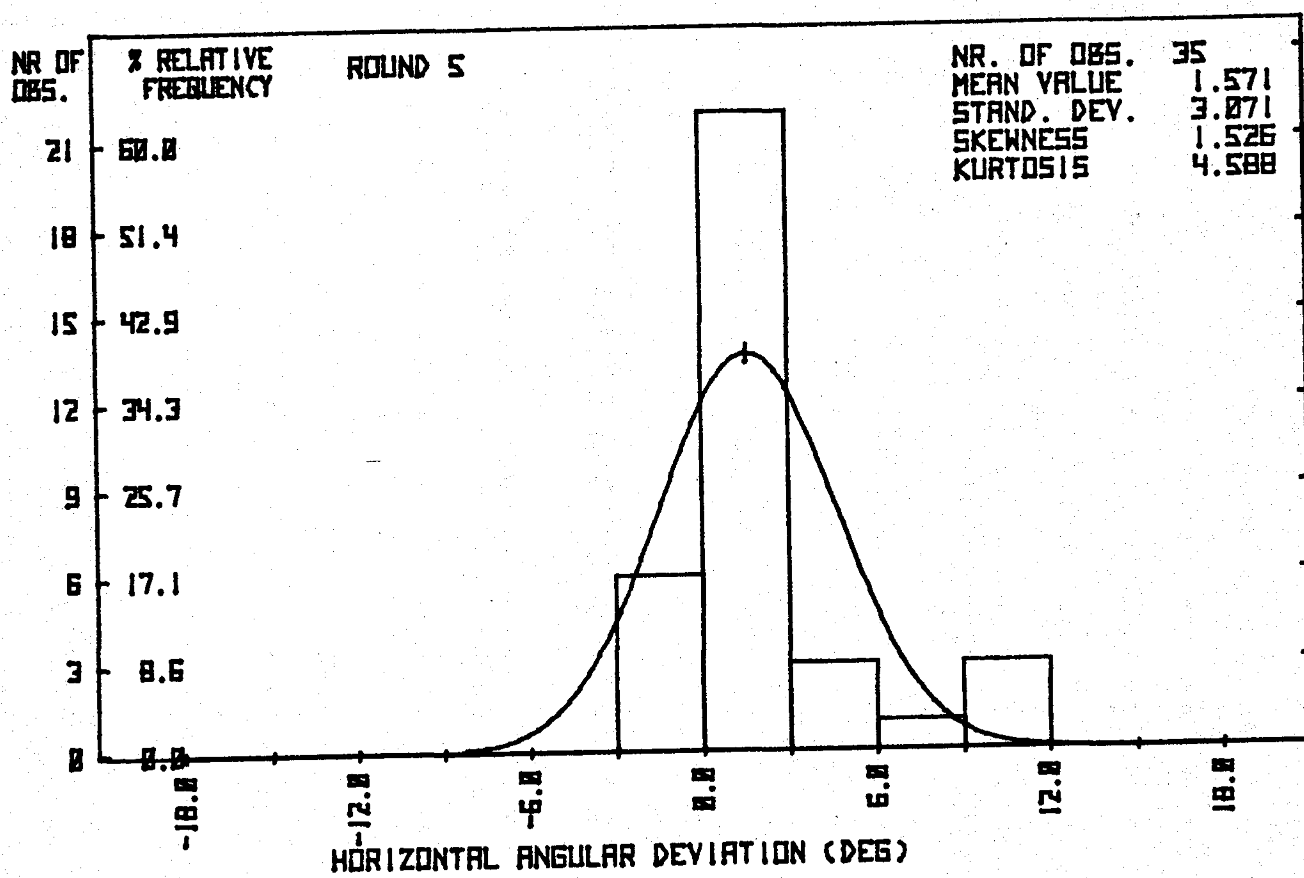
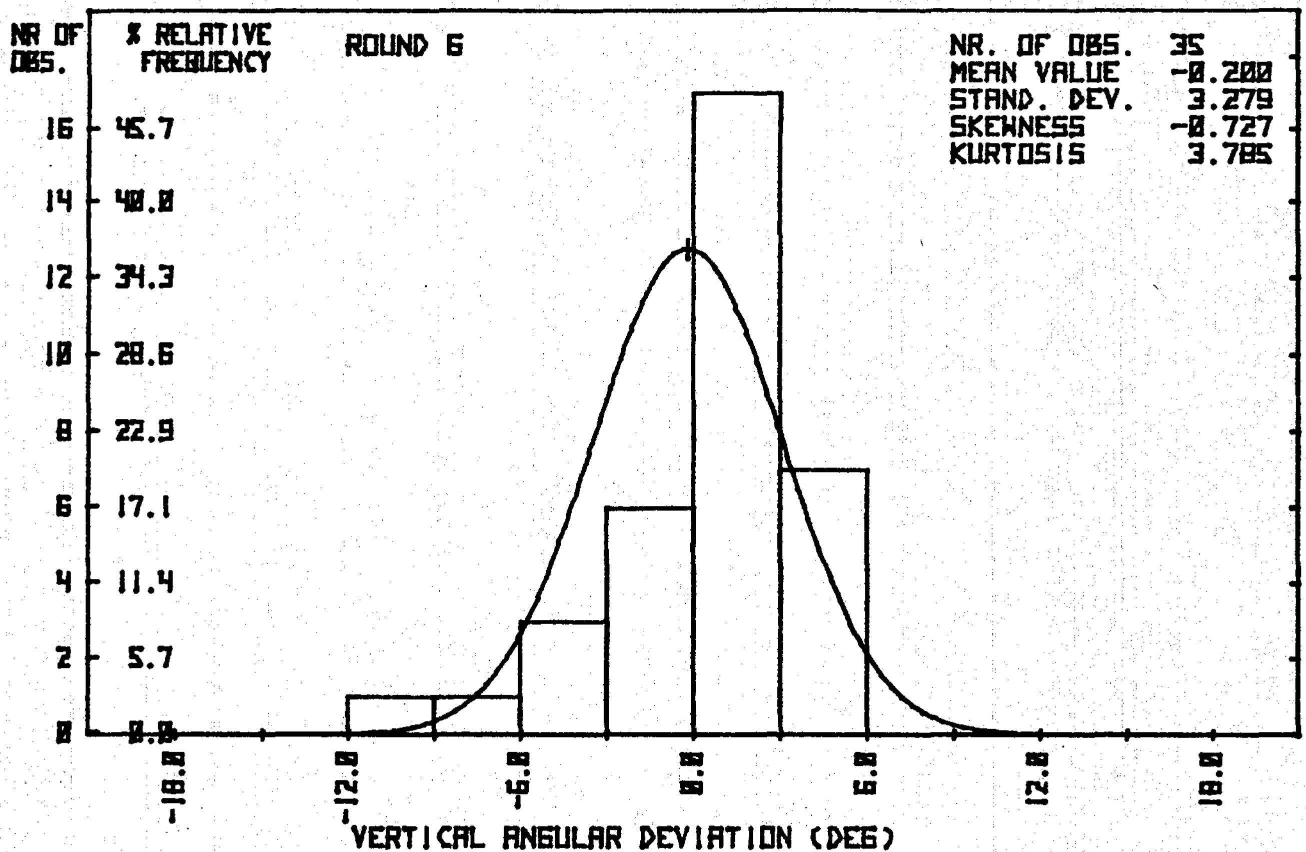


Figure C-12. Round 5 Vertical and Angular Deviations for Stoping and Cut Holes



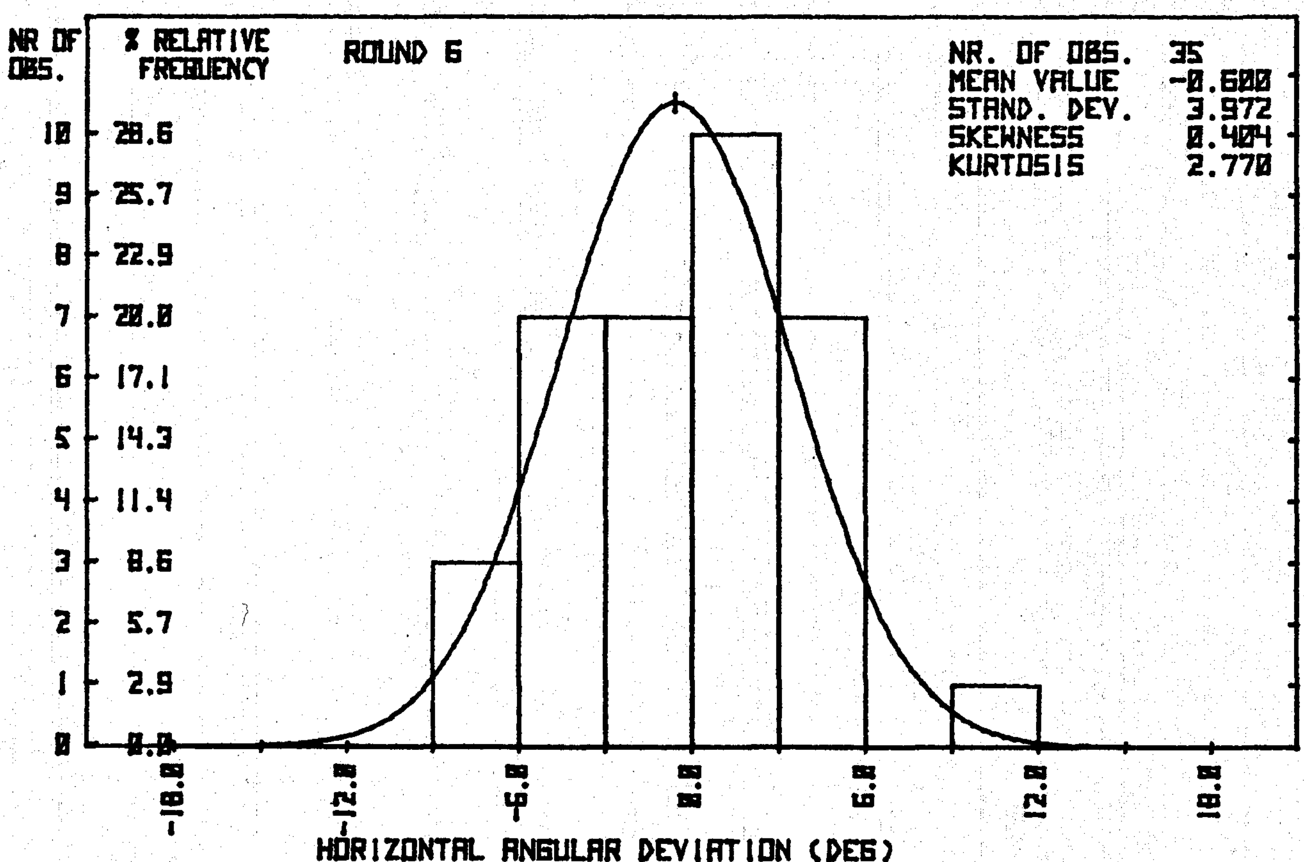


Figure C-13. Round 6 Vertical and Angular Deviations for Stoping and Cut Holes

APPENDIX D
EVALUATION OF THE ROCK CONSTANT

In the design of the initial blasting pattern, a rock constant of c = 0.4 was used. The factor c is an empirical measure of the amount of explosive used for loosening one cubic meter of rock in a specified rock geometry. The field trials that Langefors-Kihlstrom did took place in a bench geometry where the drillhole was placed in a high bench to avoid a constricted toe.[1] Blasting in brittle crystalline granite gave a c factor equal to 0.2, but in practically all other rock materials, from sandstone to more homogenous granite, the c factor was found to be 0.3-0.4 kg/m^3. Under Swedish conditions c = 0.4 is predominant in rock blasting.

The 0.4 rock constant volume was evaluated in the mine at the end of the blasting program. As there was no vertical bench geometry available, a similar horizontal geometry was used. A horizontal hole was drilled parallel to the rib to a depth of 0.6-0.7 m with a burden of 0.5 m. The hole diameter was 38 mm (1-1/2"). A 0.25-m length of sand stemming was used in all holes. Four blasts, with Tovex 210 gave the results shown in Table D-1.

Table D-1. Result for Evaluation of Rock Constant

Test No.	Charge Weight (kg)	Length of Throw for Broken Rock (m)	Comment
1	0.08	2.0	Breakage only in collar
2	0.08	2.3	Bootleg equal to 0.17 m
3	0.12	1.8	No bootleg
4	0.10	>2*	Bootleg equal to 0.10 m

* The broken rock mass hit the opposite rib.

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